

# Electrochemical and Metallurgical Industry

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## **Electrochemical and Metallurgical Industry**

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### Metallurgy in 1906

The past year, like 1905, was one of general industrial prosperity. In metallurgy we have witnessed both increased output and high prices. After the production of pig iron had reached the enormous figure of 22,992,380 tons in 1905, it appears to have passed well beyond the 25,000,000 mark in 1906. The production of gold, lead and zinc was in each case some 10 per cent larger in 1906 than in the preceding year, while the increase in copper production was 5 per cent. In the case of copper the demand has increased even more than the output, and phenomenal high prices were the natural result. Nor does it seem possible that the situation can change materially in the near future. In the electrical industries as one of the chief consumers of copper, aluminium could, of course, be used as a substitute for copper for some purposes in large quantities. But the demand for aluminium has also been greater than the supply, and the sole producer is said to be many months behind in his orders. The price of aluminium has, therefore, naturally been high; the parallelism between the price fluctuations of copper and aluminium is, of course, in no sense incidental. While it is estimated that the output of aluminium was 30 per cent larger in 1906 than in 1905, a very much larger increase is expected for 1907, since the Pittsburg Reduction Co. is making the very strongest efforts to fortify its already very strong position during the remaining two years for which it is protected in its monopoly by the Bradley patent. This situation in the aluminium industry is a most amusing example of the benefits which may sometimes be derived from a defeat in a long contested law suit.

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The increased production of metals has, of course, gone hand in hand with the specific American tendency to work on a large scale and to push the output to the utmost limit. This tendency is well illustrated by the absolute fearlessness of American metallurgists to design furnaces of dimensions undreamed of not so very long ago—just as our financiers think in millions where their more conservative European brethren are satisfied with figuring in thousands. But this boldness in design and this pushing the output of a plant to the last limit with the aid of automatic machinery will not be found alone sufficient to yield the necessary increase of production. Increased production will necessarily mean in future the treatment of more complex ores, of lean ores, of ores which were formerly considered too difficult to work. This means that metallurgical methods will have to be improved and new methods will have to be found. Above all—and we take pleasure to call attention to the admirable discussion of this phase of the subject by Dr. J. W. Richards printed elsewhere in this issue—it means that the engineer must supplant the rule-of-thumb man. This is the keynote of the whole situation.

### Electrochemistry in 1906

Since part of our last volume gives as complete and concise a record of endeavor and accomplishment in electrochemical and electrometallurgical engineering during the past year as could be made, we will touch in this note only on a few salient points. Niagara Falls is still the center of electrochemical industries in this country, and the "old" big furnace industries are flourishing—carborundum, graphite, calcium carbide, artificial emery. The most important new progress which will probably mark the beginning of a new epoch in the graphite industry is Mr. Acheson's success of making artificial soft graphite. But the natural growth of all the established Niagara industries is seriously threatened by the legislation tending toward restriction of further developments of power from the Falls. It is most unfortunate that the whole question is not properly understood by the general public. Even such a distinguished and well meaning chemical scientist as Dr. Clarke, in his able address on this matter, just delivered at the New York meeting of the American Association for the Advancement of Science, always spoke of the case as one of the American public versus the Niagara power companies. As a matter of fact, it is a case of the aesthetic sentiment (real and manufactured) of the American public versus the accomplishments of the Niagara electrochemical industries, with an immense, but by no means, fully understood economic value to the whole American public. At present we have at least the satisfaction that the decision on the most important points rests with a man of the type of Mr. Taft.

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Of new industrial developments with the electric furnace, its application in the iron and steel industry has probably attracted the greatest attention. The manufacture of ferro-alloys is an established electrometallurgical industry, but new is the growing industrial activity in this field in our country. The Willson Aluminum Co., the pioneer ferro-alloy manufacturer and the largest ferro-chrome producer in the world, has occasionally produced ferro-silicon commercially during the past year. While this is no particular metallurgical achievement, it is the first time that ferro-silicon (which has been imported in large quantities from Europe for years past) has been made on an industrial scale in this country. And others are now entering the field in not a half-hearted way. Of very much greater metallurgical interest is, however, the progress of the production of high-class tool steel in electric furnaces, both of the Héroult and of the Colby-Kjellin types. This progress is somewhat slow, but steady, and the success of the pioneer electric steel plants will act as a catalytic agent, increasing the speed of reaction between the conservative steel man and the electrochemical inventor. Of great interest are the results of the Héroult-Haanel experiments on reducing pig iron in the electric furnace, made at Sault Ste. Marie, and the results which the Noble Electric Steel Co. will obtain with the Héroult furnace in California should be carefully watched.

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A second great new achievement in electric furnace industries has to do with the fixation of atmospheric nitrogen. During the past year we recorded the success of two processes. The first is that of Frank for making calcium cyanamide in an electric furnace from calcium carbide and atmospheric nitrogen,

the first plant being in Italy. The second process, of Birkeland and Eyde, is the direct combination of the nitrogen and oxygen in atmospheric air by means of electric discharges, the successful plant being located in Norway. Without going into any details, since they have been given in full in our columns, we wish to emphasize that the Birkeland-Eyde process is an electric furnace process pure and simple, the oxidation of the nitrogen being nothing but a thermal effect. The rationale of this reaction is essentially different from the production of ozone by silent discharges through air. Laboratory work with the electric furnace has resulted in immense progress with electric-lamp filaments. First came the osmium, tantalum and graphitized carbon lamps, and they were justly considered as revolutionary achievements. But more revolutionary results were obtained last year, notably with the tungsten lamp. And now has arrived the "helion" lamp, concerning which the information is meagre though remarkable claims are made for it. The lamp is very interesting to the electrochemical engineer. The filament is described as a composite one, with a carbon core and a deposit of silicon. But a compound will result here at the temperature of a glowing filament. Have we here to do with a siloxicon or carborundum filament?

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The electrolytic industries have been no less successful. Among electrolytic fusion processes the production of aluminium naturally takes the first rank; we have spoken about it above in connection with the copper situation. The production of sodium by the Castner process is also steadily increasing. The new process of Mr. Ashcroft, whose success would result in a noteworthy reduction in the price of sodium, is now being tried in Norway. By a remarkable coincidence—it never rains but it pours—we could record in our issue of last May three different propositions for electrolytic lead smelting by Messrs. Ashcroft and Swinburne, Mr. Betts and Mr. Townsend. With respect to industries employing electrolysis of aqueous solutions, the situation has remained unchanged in the production of caustic soda and chlorine from common salt. In electroplating a gradual progress could be noticed in getting away from the rule of thumb. But of greatest industrial importance are in this class, of course, the electrolytic refining processes. Both gold and silver are now electrolytically refined in the Philadelphia and Denver mints, and the San Francisco mint is being equipped electrolytically. The lead refining process of Mr. Betts is operated commercially in three plants, one in this country, one in Canada and one in England. And though last in this summary, yet easily first in commercial importance, we have to record the continuous and splendid growth (a doubling in capacity within five years) of our large copper refineries, on which the whole industrial world on both sides of the Atlantic depends for the supply of refined copper.

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### Excess of Prosperity.

The general widespread prosperity throughout the country is certainly having most pleasant effects. Labor is well employed and fairly contented, except in large centers, where the ostentation of the newly rich spreads a trail of envy. Technical graduates are thrust into engineering positions and rapidly acquire the practical experience by the "sink or swim" method. Capital is remarkably productive. But the other side of the

shield is not so brilliant. Increase in wages is increasing cost of production. Increase in the rate of interest is hampering industrial undertaking. In short, we have a little too much prosperity. One of the most serious phases of the repleted body politic is the shortage of cars and locomotives throughout the country, but especially in the great West.

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Although the railroads of the country have placed enormous orders for rolling stock in the past two years, and as a result of these increased facilities the gross earnings are from 7 per cent to 10 per cent greater each year, nevertheless the enormous expansion in business in the past two years has made these magnificent additions to equipment look pitiful when viewed in the light of present demand. In fact, the temporary respite in business in 1903, due in part to the sentimental belief in the recurrence each decade of industrial depression was decidedly a good thing for the country. It allowed us a breathing spell in the march of progress. How serious is the car shortage on the business of the country can be seen by fact that it is quite likely that many of the big copper smelting companies of Arizona and Cananea may be forced to shut down, due to lack of coal and coke. The railroads have the right to "commandeer" any coal they need to carry on their own operations. The copper market will feel this most disastrously, for it is in a more or less supersensitive position. The floating supplies of copper are at a very low point. The demand is unprecedented. If one important source of production is partly shut off, the price of the metal, already too high, will rise to a figure so high as to throttle the electrical business to some extent, and seriously interfere with the brass business.

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The intimate connection between all forms of industry is thus so strong that the car shortage throughout the country is no less serious to the mining and smelting business than to all forms of industry. While it is acting as a brake on our too rapid expansion of industrialism, to those who feel the yoke it is a condition, not a theory. But in the gastronomic way, it is just as bad for a nation as for an individual to have "too much of a good thing."

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### President Roosevelt and Technical Training

If it be true, as the philosophers say, that true happiness consists in activity, President Roosevelt must be indeed a happy man. With "nu spelling," with killing grizzlies and giving expert advice to fathers, as advocacy, and making peace between two mighty warring nations and acting as general manager of 80,000,000 of opinionated Americans, our President certainly functionizes all his function, and should have reached the "ultima thule" of human happiness. His latest piece of literary endeavor—the recent message to Congress—deals with a most important part of our commercial world, the training of artisans and mechanics, in a thorough and expeditious manner. President Roosevelt gives due tribute to our great technical schools and to the technical research of our great universities. But he lays especial stress upon the training of apprentices and workmen by some co-ordinated scheme of manual training schools. He, however, points to the great need of such a development and the many and manifold

benefits to be expected, rather than to the exact means of accomplishing the desired results.

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In any trade there is a great necessity for skilled workers. A good mason or machinist is paid wages that make the weekly stipend of the average clerk look like the proverbial thirty pieces of our minimum coinage. And the lot of the highly paid workman from a material standpoint is far ahead of the lot of a college professor. The vast mass of our folk must of necessity be privates in the industrial army. To enable them to learn the methods and acquire the manual dexterity of their work, a complete system of manual education is necessary. This is done more and more by isolated State and city organizations. How the national government is going to further this without increasing the "centralization" of authority, already grown to an extent that would have appalled Alexander Hamilton, is hard indeed to see. Possibly some such system as that used by the Department of Agriculture in its co-operation with the several State agricultural colleges and stations will be the solution.

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The American possesses in great degree the ability to make a living. But he possesses in a slight degree only the ability of using his leisure hours with pleasure. In short, we have not developed the art of living but have developed the business of making a living. To make a young man an artisan is all right. But public opinion should do better. It should show him how to relax and to play. There has been a vast improvement in this. The diversification of industry and the development of machinery has reduced the minimum working day to eight hours. But the other hours are often foolishly spent. If each working family possessed a home with a small acreage devoted to extensive farming and with a poultry yard and a few live stock, the increase in the productivity of the United States would be enormous. This is one way to increase the efficiency of happiness throughout our land. A person's happiness is like the amperes of Ohm's law, his means of satisfying his desires divided by his wants. The current flow of pleasure should be increased by increasing in a rational manner the natural means rather than by decreasing the wants. With all our national wealth and greatness there is some foolishness about the lives of rich and poor save in isolated rare individuals. How to cultivate repose and to relax and divert the streams of energy in other lines—the proper balance between one's powers and his task, the proper distribution between one's physical, intellectual and spiritual faculties, is the greatest question of anthropology. Old King Arthur, who in his wise, practical views of life, seems to resemble our Benjamin Franklin, gave the maxim, "eight hours for work, eight hours for rest, and eight hours for play." Proper industrial training, such as Germany is following up in her slow and scientific manner, but affected by us in our effective American way, will bring about this consummation by increasing the efficiency of the workman. As usual, President Roosevelt has touched a most pertinent question of the times in a most vigorous fashion. He will arouse public opinion on this subject, fecundate an idea in many minds, and thus do more towards softening the discordant strain in the life of a nation that ought to be supremely happy.



### Rule of Thumb vs. Engineering.

The familiar Bible sentence says: "He ordereth all things by weight and measure." Granting this to be true, we have the highest justification for the principle known as "engineering," the antithesis of "rule of thumb."

The "engineer" is one who strives to perfect his work, the "artisan" works to get through with it; the first labors for intelligent love of his work, the latter "quits when the whistle blows;" the first puts "grey-matter" into his labor, the latter only muscle and experience.

Ever since man emerged from pre-historic savagery, the one who weighed chances, measured his resources and ordered his actions by intelligent foresight, has earned the place and deserved the title of "the fittest." Engaged as we all are in receiving impressions, recording observations, gaining experience, that man alone is wise who profits by this information for lay down rules for the guidance of his future conduct, for the ordering of his future actions.

The English definition of an inch is that it is "the 12th part of a foot, and equal to three barleycorns." The foot was, of course, originally the average length of the pedal extremities of a race of people and it was not only natural, but inevitable that, for example, the French foot should differ in length from the German foot, etc., etc. Not wishing to pursue this branch of our inquiry to too great and perhaps tiresome lengths, we may as well recite at once a striking and suggestive fact, whose connection with our argument may not be at once apparent; viz., the French word for an inch is "le pouce," which is also the French word for "the thumb." The connection implied is, of course, that an inch, in France, at least, and probably in England also, was originally the length of the second joint of the thumb, which varies less in different people than would at first thought be imagined.

In the absence, therefore, of a proper foot rule marked into inches and fractions, it was customary at one time to measure lengths of small dimensions with the thumb joint and so get an approximation of the number of "thumbs" in a given length. It would be ill-advised, if not conceited, for us to poke fun at or wax sarcastic about the absurdity of such practice, for the "rule of thumb" was a valuable practice when compared with what it displaced—no rule at all; and just as imperfect laws are better than total lawlessness, so the "thumb rule" marked a decided advance towards the arts and sciences.

Somewhere later in time than the period we have been considering, measures of length accurately compared with an arbitrary standard began to multiply and to displace the "rule of thumb." Artisans could then begin to work with exactness, the carpenter could plan and scheme ahead accurately by measure, the smith could forge articles accurately to any required stated size, the gentleman could order from his tailor trousers so many inches long, without allowing for the different lengths of thumb of his sartorial "churls." This period having arrived, and this stage of advancement in the arts and sciences having been reached, we are now prepared to let loose the vials of our criticism, or, perhaps, indignation, and with perfect justification, upon the unlucky wight, who, either from inborn "cussedness" or acquired laziness *continues* to afflict humanity with "the rule of thumb." Here is the unpardonable sin: the accurately divided rule has come, and the confirmed sinner sticks to his "thumbs" and "barleycorns."

Having dragged the reader through this somewhat misty prologue, it is becoming that we state exactly what we mean to talk or preach further about. It is just this: The difference between "rule of thumb" and accurate calculation, the wide gulf separating the hit-or-miss cut-and-try methods of the ancients and the scientifically-informed practice of the modern technologist. Not that we are going to hold up the former to ridicule or the latter to extravagant adulation, but we wish to discuss some of the features of modern technol-

ogy which illustrate the great change which has taken place and the still greater changes which are possible.

To take an illustration from civil engineering: Spanning the Firth of Forth, a few miles above Edinburgh, stands the greatest structure of the present time. With clear spans of 1,700 feet, capable of carrying the heaviest trains, the strains and stresses in each of its thousands of members all accurately calculated beforehand, here is a monument to engineering skill absolutely unthinkable as a product of "rule of thumb."

To take a mechanical illustration: Down in the hold of a monster steamship, compressed within the smallest possible working space, are engines developing, night and day, 40,000 hp. Every nut, bolt and screw has its assigned place, every accurately calculated and measured part its function, every unequal strain is balanced, economical operation provided by a complex system of quadruple expansion, every one of the thousands of parts all working in complete harmony and under exact control. The triumph of mind over matter, you say; yes, but better as an illustration of the infinite superiority of "engineering" over "rule of thumb."

Look once more at the electrical engineer and his achievements. A huge dynamo spins like a top on a shaft 160 feet deep, and develops in the smallest compass imaginable 10,000 hp. The diameters of the armature wires are accurate to the thousandth of an inch, the hundreds of turns of wire are neither one too few nor one too many, the very phases of the pulsations of the electric waves are mapped and studied out so as to bring out the maximum efficiency. Soft iron from Norway, mica from Canada, rubber from Paraguay, copper from Arizona, shellac from China, steel from Pennsylvania, are combined with the most consummate skill and accurate calculation into a huge giant, which works day and night with almost the regularity of a planet. "Rule of thumb" could never have conceived of such a product; only "engineering skill" has made it possible.

One Art and one Science still bear much of the stigma of "rule of thumb:" the Science of Chemistry and the Art of Metallurgy.

Chemistry remained Alchemy until the invigorating breath of modern methods commenced to blow away the mists of mystery which enveloped it. The evolution has been long and laborious. Conservatism in the managers combined with stubborn old-fogyism in the workman, have conspired against enlightening methods to keep the industry in the rule-of-thumb rut. Germany was the first country to break away into scientific and engineering chemistry, and the lead thus gained has been maintained to this day. The Britisher, workman or manager, is the most conservative of human beings, and his backwardness in following Germany's methods is the chief cause of the industrial difficulties under which the British chemical industry now labors.

Metallurgy is still rule of thumb in many countries, such as Central Africa (production of iron), China (production of lead and mercury), Straights Settlements (reduction of tin), where aboriginal methods still largely survive. But, the aboriginal spirit is preserved in many other so-called civilized countries; it persists among the Cornish and Saxon tin workmen, the Derbyshire and Carinthian lead smelters, the Welsh and Mansfield copper refiners, the Alaskan and Siberian gold winners, the Missouri and the Silesian zinc distillers, the Almaden and Idrian mercury mines, the Eastern Pennsylvania as well as the Lithuanian blast furnaces.

Among all of these, and others too numerous to mention, are to be found that reverence of tradition, that willingness to keep plodding in the rut, that indisposition to apply the discoveries of modern science, that ignorance of what modern science has disclosed, which keep them groping in the semi-darkness of medievalism, while their more intelligent neighbors are running away from them like an express train distancing a stage coach.



To be more specific, and therefore more to the point, let us catalogue the principal items of this indictment, the details of modern scientific discovery whose neglect is the reason for the preceding Philippic. The broad principles involved are simply—neglect to order "all things by weight and measure;" in other words, neglect of the only means by which *intelligent control* can be exercised over chemical and metallurgical processes, or, in still other words, preference of *rule-of-thumb* to *engineering skill*. These principles can be sub-divided, for further consideration, into the observation and utilization of

- (1) Weights.
- (2) Compositions.
- (3) Volumes.
- (4) Pressures.
- (5) Thermometry.
- (6) Calorimetry.

(1) The first and probably most important method of control, is that of *weight*. The use of the scales, to determine how much material is being used and how much produced, and to regulate the items of charging and discharging a furnace or other apparatus, is so nearly universal that it is with difficulty that we can imagine the chemist or metallurgist doing any work without them. Yet there was undoubtedly a time when both chemical and metallurgical operations were conducted without a knowledge of the weights involved, when it was not known that more pounds of ore must be used than pounds of metal are obtained, and when the most delightful degree of irresponsibility must have been felt by the workman as to the output of his process.

We can feebly imagine the tremendous "kick" made by the first Tubal Cain who was furnished with a pair of scales and told to record the weights of all he used and all he produced. How he would execrate the useless labor involved, condemn the "powers that be" to the Styx for increasing his labor so unnecessarily, and even possibly damage the obnoxious weighing machine surreptitiously if he got the chance.

Yet, such behavior sounds too familiar to be altogether confined to ancient history. Is it not true, that even in our day, the scales are not used as much or as carefully as they should be? To put the indictment in a mild form, are not weights taken too little in detail? For instance, the coal used by a furnace, running continuously, is put into a bin and the amount used per week is accurately known; but, if the weights taken out of the bin were determined for each shift of workmen, it might be determined whether the night shift was as economical of fuel as the day shift or not.

In metal-melting establishments, fluxes are often used much more lavishly and extravagantly than they should be; a careful account of the weights used by each melter for each melt would often illuminate matters. All sorts of extravagancies and negligencies may pass for years unchecked if not controlled by careful weighing, carried out in great detail and for each individual unit. The cost of this slightly greater attention to detail will almost invariably be several times repaid by the increased information acquired and more intelligent control thereby made possible.

(2) The question of compositions is the purview of the analytical chemist. How slowly the value of this information is being recognized! Next in importance to the amounts of material being used or produced is their quality, and here rule of thumb has reigned supreme until almost our own day. This ore looks certainly better than that, this fuel is finer than that one, as anybody can tell; that slag is all right, you can't see any metal in it; this pig iron is first quality, just look at the fracture—and so the rule-of-thumb man keeps on delivering dicta, drawn from experience, most of which are right and some of which are egregiously wrong.

The greatest benefactors of chemistry and metallurgy in the nineteenth century were those who developed analytical chemistry into a practical art—such as Berzelius and Fresenius, and those who led in its practical application, such as Bell, Bes-

semer, Muspratt and Lunge. It is surely unnecessary to give examples of what the analytical chemist has done for all branches of applied chemistry. He has been the principal factor in modern scientific control of all these operations; and the logic of this circumstance is so strong, that the chemist in the works laboratory has become the natural candidate for the superintendency of the plant.

The chemist in the laboratory is primarily a machine to do analytical work; but, granting him only moderate ability, he soon acquires incidentally such intimate insight into the *rationale* of the works' operations that he becomes indispensable out in the works as conductor of those operations.

The chemist should need no recommendation to modern technologists; they should all know by this time how hopeless it is to get along satisfactorily without him. But, there are still some adherents of rule of thumb who think they can; there still exist some foundrymen who believe they can tell good pig iron when they see it (but they cannot), steel workers who can tell the quality of steel from its fracture (they are often fooled completely by modern steels), blast-furnace men who can distinguish good coke from bad by the looks (retort-oven coke contradicts their experience), and so on, through all the range of chemical and metallurgical technology.

The modern way to get scientific control of this question of *quality* is to use the chemist, with his methods of examination—analytical, microscopic, experimental—and so keep going a works laboratory. If it does not pay for itself several times over, it will be through human imperfection in the laboratory or in the management of the works, but not to defect in the principle involved.

In any but large works the chemist will also be the physicist of the plant. He will be looked to to make those physical determinations of volumes, pressures, temperatures and heat distributions which are so badly needed, but so little employed. Incidentally, it may be observed that he is usually, by education and training, more fit to do this work than the mechanical engineer, who is often asked to perform it.

(3) The use of instruments to determine the volumes of gases involved in operating any process, is an improvement which tends away from rule of thumb and towards accurate control. The use of the anemometer to determine the volume of air-supply to a furnace, of the Pitot tube to measure the gas supply in a closed tube, of the draft gauge, plain or multiplying, to estimate resistances to flow and in general all the methods by which the volume and velocity of gas or air currents may be made known, are first aids to the scientific chemist or technologist.

Using forced draft it is possible to measure the delivery of a fan and adapt it exactly to the feed of coal dust so as to produce nearly perfect combustion; many a boiler gives poor results because twice as much air is used as is needed to burn the fuel, causing high chimney losses.

When managers realize, as they should, that such information means, if properly used, stopping of large leaks and obtaining of higher efficiencies, they will be quicker to avail themselves of the more accurate control thereby assured.

(4) In some operations, pressure plays an important part in the operation and should be accurately controlled by properly calibrated instruments. The back-pressure in a blast furnace is known indirectly by the increased work thrown on the blowing engines, but it is much better known by a suitably-placed gauge. An unexpected fall in pressure between two points on a blast main may indicate too small a main or an obstruction or high friction, which might be cured by making the interior smoother. If pressures are involved in any operation, an active use of the pressure gauge is a constant *desideratum*.

(5) Thermometry tells the intensity of heat effects and is being less and less neglected as the methods of pyrometry are being more and more perfected. Nothing is of more importance for any furnace process, than that the temperature conditions should be kept constant or be reproducible at will, and

the only sure control of this is instrumental determination of the temperature.

The human eye, when highly trained, can determine temperatures between a low red and a bright yellow with a considerable degree of accuracy, providing the eye is not fatigued and the possessor of it has a liver in good working order. Good instruments, however, are more accurate, and are less susceptible to fatigue or disorders, while they can be made to register at a distance and even to give a continuous record.

All these improvements have tended to remove empiricism and to replace it by accuracy, to take the control of the operation out of the hands or skill of the workman and vest it in the real manager of the work. Finally, it permits of high-priced skilled labor being supplanted by faithful, but less experienced and lower-priced men, all tending to greater regularity of output and lower cost of production. One of the most satisfactory developments in the last ten years has been the great increase in the number of reliable pyrometers and in their general use. That user of furnace processes who is not equipping his plant with pyrometers is surely allowing his competitor who does, to gain a great advantage.

In the metallurgy of iron, pyrometers have, until recently, been used only on the hot-blast main of the blast furnace. They should be used at the top of the furnace, at entrance to the stoves, at chimney flues and to control temperature of pig iron and slag. They should be used on mixers, puddling furnaces, open-hearth furnaces, bessemer, casting ladles, reheating furnaces, annealing furnaces, cementation and malleableizing furnaces, and for hardening and tempering.

In the metallurgy of copper they should be universally used to control the roasting operation and matting furnaces. In the metallurgy of zinc, the roasting and distilling furnaces will be under better control the more they are investigated pyrometrically.

Too high temperatures are often as greatly destructive as too low temperatures are inefficient, and both can be avoided or controlled by the pyrometers. Space forbids the cataloguing of even a fraction of the places and processes wherein pyrometry should be used more largely; it is one of the principle methods of controlling an operation in the hands of the modern technologist.

(5) Lastly, calorimetry measures quantities of heat and tells in what directions the furnace is expending its energy. One boiler may evaporate 8 pounds of water per pound of coal, and another 10 pounds, but when we learn that the heat in the steam is only 40 and 50 per cent, respectively of the calorific power of the fuel, we first become aware of the great room still left for improvement.

Again, a steel-melting crucible delivers as useful heat in the melted steel only some 3 per cent of the calorific power of the coke burned; an open-hearth furnace 25 per cent; a cupola 35 per cent; an electric furnace 50 to 75 per cent of the heat value of the electric energy expended. In all these cases, information of the highest value to the metallurgist can only be obtained by calorimetric measurements and calculations based thereon.

When a furnace process is thus analyzed thermally and the distribution of its available heat energy tabulated, so much to radiation, so much to imperfect combustion, so much to chimney loss and so much (usually *so little*) usefully applied, there becomes possible a true esoteric view of the operation.

Only on such broad grounds can the process be thoroughly understood in all its relations and such information opens the way to economies often unthought of before. We therefore plead earnestly for the calorimeter as one of the chemist's and metallurgist's most valuable, and perhaps most neglected, auxiliaries.

The conclusion of our argument is that technologists, in whose hands lies the application of science to human needs, should assist their endeavors and perfect their methods by the use of all the means of attaining accuracy of control

which science furnishes them; that they should use and profit by the instruments of weighing and measuring masses, volumes, pressures, temperatures and heat; that they should use the analytical chemist more than ever before; and, finally, that by these means only can they reach the goal of the highest attainable efficiency.

The rule-of-thumb man guesses: the engineer calculates.  
JOS. W. RICHARDS.

### The Iron and Steel Market.

The year 1906 has been by all odds the most prosperous one the American iron industry has ever enjoyed. In a few departments profits per ton have not been as great as in some other years, but tonnage has been very much larger. No previous year can at all compare in point of tonnage with 1906 except the year 1905, while the general level of prices was higher in 1906 than in 1905, although, except in pig iron, they did not reach an unhealthy level.

#### PRODUCTION IN 1906.

The total production of pig iron may be estimated as more likely to be over than under 25,300,000 tons, against 23,000,000 tons in 1905, a gain of an even 10 per cent. The year 1906 closes a decade in which but one year has failed to break all previous records for pig iron production. In 1897 and 1898 the gain in production was not accompanied by a sufficient gain in demand, and these two years saw the lowest prices ever recorded in the American iron trade. They will doubtless prove to be the lowest unless the monetary standard changes. A part of the advance in prices since then is really due to a lowering in the real value of the gold dollar. In 1899 demand suddenly recovered, and with a heavy gain in production prices nevertheless soared; 1900 saw a reaction in prices, with a slight gain in production; in 1901 and 1902 prices steadily rose, despite heavy increases in production; towards the close of 1903 a severe reaction set in, but the decreased production in the fourth quarter did not prevent the calendar year from showing a slight gain in production over 1902; the full force was felt in 1904. From towards the close of that year to the present time production and prosperity have been increasing hand in hand. Briefly, the course of pig iron production has been as follows: In 1897 a trifle under 10,000,000 tons, then a steady increase to about 18,000,000 tons in 1902 and 1903, a decline to 16,500,000 in 1904, and a gain to 23,000,000 in 1905, and to over 25,000,000 in 1906.

While pig iron is an excellent yardstick with which to gauge the production of commercially usable forms of iron and steel, it is a fact that production of the latter has increased somewhat more rapidly in recent years than production of pig iron. A careful estimate of the finished forms of the iron and steel trade, *i. e.*, of iron castings, steel castings and finished rolled iron and steel forms, shows that at present the tonnage is about equal to that of pig iron, the necessary loss of pig iron being fully made up by the use of scrap and of a small quantity of iron ore in the open-hearth process. The basic open-hearth process has in a very few years become a close second to the Bessemer in point of tonnage, and uses large quantities of scrap. The various processes for converting scrap into wrought iron, including puddling, knobbling, busheling, faggoting and piling on boards, decadent for many years, have taken on new life in the past three years. The iron mills, although using less than half a million tons of pig iron a year, are producing more than 2,000,000 tons of rolled products.

#### PRICES IN 1906.

A heavy buying movement at the close of 1905 left the iron trade but little to do in the early part of 1906 but get out tonnage. Prices were very steady—*i. e.*, for so fickle an industry—during the first half. Bessemer and basic pig iron and nearly all finished steel products were uniform in price throughout the half year. About March 1 iron bars had a

sharp drop, and about April 1 Northern foundry iron declined somewhat; in April, steel bars were shaded by one prominent producer, and in June occurred a sharp drop in Southern pig iron to \$13, Birmingham.

Immediately after July 1 the whole complexion of affairs changed. In July foundry pig advanced slightly on light buying. In the first half of August the trade was startled by large purchases of Bessemer pig iron for delivery in the first half of 1907; in the second half of August foundry pig iron underwent the same movement. Thereafter all grades of pig iron advanced rapidly to famine prices. With the middle of September began a series of moderate advances in finished steel products, involving, by the end of the year, every finished steel product of importance except structural shapes and standard rails. Standard rails are a law unto themselves, having been maintained through thick and thin at \$28 since the spring of 1901. Light rails, usually cheaper than standard rails, obviously from commercial rather than technical reasons, advanced to \$33. Shapes had been advanced in August, 1905, at a time when other steel products were not being advanced, which probably accounts for their being an exception in this case.

A number of these advances have been in lines where it is the accepted trade usage for the mills to advance prices when a large tonnage has been entered, in order to protect contracts and induce specifying. Notable instances of these are merchant steel pipe and wire products. In these it may be possible to establish a lower level without disturbance to the general price fabric, but when pig iron declines, as ultimately it must, serious disturbance can scarcely be avoided. It is true that finished products have not been advanced by any means in proportion to pig iron, and the latter ought, therefore, to be allowed to decline by itself, but this theoretical argument was put forth in 1902 and 1903, and the outcome proved that the business world would not follow the theory. Pig iron reached up and dislodged finished steel prices.

#### THE CURRENT SITUATION.

The new year opens with substantially all the first-half pig iron output sold, and modest commitments being made for second half. More than half the year's rail output is sold, and the tonnage would undoubtedly be much larger were not many railroads waiting on open-hearth rails. Plates, shapes and steel bars are contracted for nearly the whole year, while in plates and bars specifications are actually assured for nearly six months' operation. In pipe, wire, sheets and tin plates absolutely certain business is booked for from three to six months. In all lines specifications are coming in freely.

#### PIG IRON.

It is useless to attempt in a brief space to give pig iron quotations accurately, since time of delivery and other considerations are all important. Southern No. 2 is about \$23.00, Birmingham, for prompt, \$21.00 to \$22.00 for second quarter, \$18.50 to \$19.00 for third quarter only, and \$18.00 to \$18.50 for third and fourth quarters together. Northern No. 2 foundry, at valley furnace, is \$25.00 for prompt, \$22.50 to \$23.00 for second quarter, \$21.50 for third quarter, and \$21.00 for second half. In these prices there is a tapering off, as to time of delivery, which will do much towards lessening the influence of the inevitable decline as production increases, since such an increase is well recognized as in prospect. Bessemer pig iron is \$21.00 to \$21.50 at furnace for second half.

#### PRICE ADVANCES.

Since last report the following advances have occurred:

Plates, \$2.00 a ton to 1.70c., Pittsburg, for tank quality.

Splice bars, \$3.00 a ton to 1.65c.

Wire products, \$2.00 a ton to \$2.00 a keg for nails and 1.85c. for plain wire.

Light rails at Pittsburg, \$1.00 a ton to \$33.00 for 25 to 45-pound sections, in large lots; \$34.00 in single carloads.

Merchant pipe, two separate advances of 1 point each.

#### FINISHED MATERIAL.

We repeat former quotations on the following:

Structural shapes on basis of 1.70c. for beams and channels, 15 inches and under.

Merchant steel bars, 1.60c., base.

Sheets, 2.60c. for black and 3.65c. for galvanized, No. 28 gauge.

Tin plates, \$3.90 for 100-pound cokes.

Sheet bars are about \$30.00, Pittsburg. Billets are very scarce and quite irregular in price, being nominally about \$29.50 to \$30.00 for Bessemer, and \$32.00 to \$33.00 for open-hearth.

#### Faraday Society

##### ELECTROLYTIC PRODUCTION OF BLEACHING LIQUOR.

At the November meeting of the Faraday Society two papers were presented dealing with electrolytic hypochlorite. Mr. W. POLLARD DIGBY discussed "the depreciation of electrolytically produced solutions of sodium hypochlorite." This paper deals in the first place with depreciation taking place in bottles of various colors in which dark amber bottles gave the best results, the loss in 1,817 days being about 40 per cent for a solution containing 4.216 grms. of available chlorine per liter. The corrosive action of hypochlorite solutions upon various metals is then discussed, and the depreciations due to graphite, copper, zinc, lead and iron plates immersed in such solutions are set forth for a period of 480 hours. A much greater depreciation takes place, due to galvanic action, when two dissimilar metals immersed in the liquid are connected by an insulated wire; the paper gives records in the case of twenty-one different couples. When iron is present as one metal in such a couple, the depreciations are generally greater than for any two other metals.

As regards the area of the exposed surface in the cases of single metals immersed in hypochlorite solutions, it was found that doubling or trebling the surface doubled or trebled the rate of depreciation. Brief reference is made to the value of protective coatings as applied to metal couples, but no conclusive results were obtained.

As it is a matter of difficulty to paint evenly the inside of small plates, and, still more, of pipes, the writer deprecates reliance upon insulating materials for the protection of any vessel in which hypochlorite liquids are stored, or of any metallic piping in contact with hypochlorite liquids. When once the paint breaks down, corrosion of the metal will follow at a rapid rate. Accounts are also given of the value of bitumen as a protective agent for wood exposed to hypochlorite solutions.

In conclusion, the author urges that storage, transport and methods of conveyance from place to place, or even from one part of a factory to another, are details meriting attention. Certain conditions of storage may clearly be regarded as fatal to stability, particularly those in which the galvanic action of dissimilar metals is likely to arise.

Mr. CHAS. V. BIGGS then presented a paper on "The Hermite Electrolytic Process at Poplar." This is a municipal plant, making a solution containing about 4.5 grams of available chlorine per liter, for use as a disinfectant in the boroughs.

The system adopted is to mix a certain quantity of fluid in an elevated tank, and then to allow this fluid to flow through four double troughs or cells, placed one above the other so that the liquid descends continuously by gravity. Each trough is divided laterally by a partition, and in each of the two divisions five distinct "elements" are suspended. The positive plates are of thin platinum wire wound upon slate slabs and the negative plates are of zinc. There are thus four troughs, each containing ten "elements," or forty cells in all. After passing through the successive cells the liquid discharges into a carboy. Sodium hydroxide is used as a preservative; it flows drop by drop into the carboy as it is filling, and serves to neutralize free



hypochlorous acid. As the liquid passes through the troughs it is subjected to the action of a current of 15 amps. at 230 volts, being 5.6 volts per cell.

The tank is charged by placing in it 100 liters of a saturated solution of sodium chloride and 20 liters of a saturated solution of magnesium chloride. To this is added as much water as will bring the whole up to 840 liters. The rate of flow through the apparatus is 3 1-3 pints per minute.

Curves are appended showing (1) the rise in value of the solution as it passes through the cells, amounting to about 1 grm. per liter for 10 cells; (2) efficiency in grms. per B. T. U. for this plant, which is highest at a current of 16 amps. Above this amperage energy is wasted in heat, and below it the flow becomes too high.

A number of tests of samples of the liquid made are given. Liquid made during the warm weather of the exceptional summer of 1906, at a temperature of 104° F., was found to have lost only 0.1 grm. available chlorine per liter in six weeks. Several samples of liquid made before the conditions of the plant (as to mixing, rate of flow, etc.) were very fully determined, showed losses of 0.1 and 0.2 grms. per liter in six months; the higher strengths (4.8 to 5.3 grms. per liter) losing 0.5 grm. in six months.

The author concludes that the magnesium hypochlorite, as made at Poplar, is sufficiently stable for practical purposes, and that it could be made in a warm climate without necessarily rapid deterioration. The warm weather specimens quoted had been kept in ordinary dark glass bottles, merely corked.

Messrs. J. B. C. Kershaw, R. S. Hutton, L. A. Senart and W. Defries joined in the discussion. Mr. F. W. Alexander gave complete details regarding the cost of running the Poplar plant. The plant has been producing an average of 200 gallons a day of 0.4 grm. solution since January last, at a total cost for current and material of under \$165. Two laborers suffice to watch the plant and bottle the solution. The defects of the original Hermite process have been successfully overcome. They have had no trouble with non-conducting films on the zinc cathodes. Non-stability is caused by the presence of undecomposed salt in solution. (See also p. 26.—Ed.)

#### STORAGE BATTERIES AND THEIR ELECTROLYTES.

At the December meeting a paper by Mr. R. W. VICAREY on this subject was presented. It is a continuation of one read by the author in July, 1905, and deals chiefly with some of the problems involved in the manufacture of accumulators, particularly as regards the effect of nitrogen and other impurities introduced consciously or by accident in the process of manufacture. Nitrogen compounds are often used, and advisedly so in the opinion of the author, for assisting in the oxidation of the positive plates, and in such cases it is essential that every trace of nitrogen be removed before the cells leave the works, in spite of the difficulties, which are insisted upon, involved in its complete elimination. The evils attributed to nitrogen, both after complete formation and in the latter stages of the forming processes, are described in detail.

Among other troubles the author has observed the formation of an irreducible and unoxidizable sulphate of lead ( $2\text{PbSO}_4 \cdot \text{PbO}$ ), a detailed study of the effects of which are promised for a future communication. The evil effects of nitrogen are greatly accentuated by the presence of other impurities—notably iron, an element likewise very difficult to eliminate—and experiments are described illustrating these effects, which become more serious with decreasing current densities. Researches of Peters, Elbs and Bell are quoted in support of the author's main contentions.

Mr. H. L. Joly criticised in detail many of the author's contentions. He could not accept the extreme "ammonia" theory claimed by the author, who had not furnished sufficient definite and reliable data to found hypotheses upon. He would like to have had an account of the physical constants of the "new" lead sulphate.

#### THEORETICAL PAPERS.

Two papers were presented by Dr. A. C. CUMMING. This first paper was entitled "Contributions to the Study of Strong Electrolytes," and its first half dealt with the elimination of potential due to liquid contact.

Certain solutions have the property of reducing the potential due to the contact of two solutions, and potassium chloride has been used for this purpose. In most cases a saturated solution of potassium chloride does not remove all the diffusion potential; indeed, if the solutions in the cells be strong, it only removes a small part. This property of removing more or less of the diffusion potential depends on two factors in the connecting solution: firstly, the positive and negative ions must be of equal velocity; and, secondly, the concentration of the connecting solution must be high compared with the solutions in the cells.

The author suggests a saturated ammonium nitrate solution as that which fulfils these two conditions better than anything else at present known, and shows by experiments with different cells that this is the case.

A curious point was noticed when two cells containing the same solution, but of different concentrations, were joined by a third solution of the same substance. Whatever the strength of this third solution, the observed potential remained the same as when the two cells were directly connected with one another. This could have been predicted from Nernst's theory.

The second half of this paper gives the results of measurements of the e. m. f. between silver nitrate solutions of various strengths from 1/1000 normal up to 1/2 normal, three kinds of silver electrodes being used. The results prove that for silver nitrate the e. m. f. gives the same measure of the ionic concentrations as is obtained from the conductivities, and therefore support the view that the conductivity gives a true measure of the ionic concentration.

A second paper by Dr. A. C. CUMMING deals with "The Electrochemistry of Lead." The tendency of any *ic* ion to be transformed into an *oms* ion with a lower valence may be measured by means of an oxidation electrode. A solution containing plumbic lead was obtained by dissolving  $\text{PbO}_2$  in nitric acid, and the amount estimated analytically, while any desired strength of plumbous lead was obtained by addition of ordinary lead nitrate. The potentials were measured with a lead peroxide electrode, against a N calomel electrode, with a saturated ammonium nitrate solution as connection. The mean value for  $\text{Pb}^{++} \rightarrow \text{Pb}^{+}$ , i. e., the tendency  $\text{Pb}^{++} \rightarrow \text{Pb}^{+}$ , was found to be + 1.82 volt against the hydrogen standard electrode. The value for  $\text{Pb}^{++} \rightarrow \text{Pb}$  was also redetermined, and found to be - 0.137 volt, so that the value for  $\text{Pb}^{++} \rightarrow \text{Pb}$  is + 0.84 volt.

The results in general prove that lead in the tetrad form is a highly electropositive element, and also direct attention to a curious difference in the behavior of sodium and potassium nitrates towards lead nitrate.

## CORRESPONDENCE.

### Electric Smelting of Iron Ore.

To the Editor of *Electrochemical and Metallurgical Industry*:

SIR:—In your November issue you stated that the Northern Californian Power Co. were engaged in establishing a plant at this place for the reduction of iron ore by the Héroult process. In this you were mistaken, as Mr. H. H. Noble, president of that company, is solely interested in the undertaking, and the only interest that the power company has in the subject is in the creation of a large customer for electric power.

Work is actively progressing at this place in preparing ground for the plant, all of which has been ordered and will be installed early next year.

NOBLE ELECTRIC STEEL CO.  
Héroult-on-the-Pitt, P. O. Baird, Shasta Co., Cal.

## The Girod Ferro-Alloy Works and the New Girod Steel Furnace.

By DR. R. S. HUTTON.

A recent visit to the chief works at Ugine, in Savoy, of the Société anonyme Electrometallurgique. Procédés Paul Girod and the courtesy of the director of this company in supplying photographs and information have enabled the following brief account to be prepared for the readers of *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*. The company at the present time possesses three important works, viz.: at Ugine, in



FIG. 1.—THE GIROD WORKS AT COURTEPIN, SWITZERLAND.

Savoy, at Courtepin and at Montbovon, in Switzerland. A total of 18,000 hp. is in use, but the harnessing of further power is rapidly progressing, and shortly some 45,000 hp. will be available.

The history of the development of these large works follows the lines which have been so characteristic of many of our most important electrochemical industries. Starting in 1898 with a small experimental plant of only 28 hp., Monsieur Paul Girod attacked and gradually overcame the great difficulties underlying the production of high-grade ferro-alloys. In 1899 a works employing 1,000 hp. was started at Albertville, and with accumulating experience still more ambitious schemes were planned and executed. The great success which this company has had may, doubtless, in some measure, be attributed to the foresight of its director and to the fact that this early experience was quickly applied so soon as the development of rapid-cutting tool steels and other special steels created a demand for ferro-alloys.

At the end of November, 1903, the works at Courtepin (see Fig. 1) were started with 1,800 hp., which has been rapidly increased to 5,000 hp., the power being obtained at 16,000 volts from the Fribourg Cantonal Government. In the same year the water rights of the falls of the Arly were purchased, and the installation of the power plant of 8,500 hp. at Ugine was completed in a remarkably short space of time, being available for use in December, 1904.

### INSTALLATION AT UGINE.

The French Alps offer many surprises to those unfamiliar with the remarkable hydro-electrical installation in this part

of the European continents. The valley of the Arc in its descent from Modane to Saint Michel with its four large aluminium works and numerous other smaller factories; or the valley of the Romanche, near Grenoble, with carbide and ferro-alloy works, both form centers of electrochemical industry which are already comparatively well known. Even in these cases the rapid changes which have recently been effected and the enormous developments which are in progress cannot fail to impress those who have closely followed the trend of electrochemical work in this part of the world.

In the beauty of its surroundings and in the evidence which is forthcoming of extensive and successful development, the works at Ugine (see Fig. 2) are, however, almost unsurpassed. Situated at the foot of snow-clad mountains in a peaceful valley, almost luxuriant in its vegetation, the contrast with the more usual surroundings of works connected with the iron and steel industry is very marked. The fall available is 125 meters, the water being conveyed from the *barrage* through a tunnel of 3 kilometers, and then by two steel conduits each of 1.32 meters diameter and 550 meters length.

The power house, 120 meters long and 10 meters broad, shown in Fig. 3, contains nine 600-hp. and nine 300-hp. Neyret-Brenier turbines, driving six 600-hp. and nine 300-hp. continuous-current machines, and three 600-hp. alternators. The furnace house runs parallel with that in which the machines are installed, and is equipped with a large number of special furnaces of the "smothered-arc" type, provided with automatic regulation, efficient ventilation and all the equipment necessary for successful and continuous working.

Auxiliary plant, such as crushing and grinding machines and mixers, is also plentifully available.

### PRODUCTS OF THE WORKS.

The present annual output of the three installations, so far as the chief products are concerned, may be summarized as follows:

- 5,000 tons ferro-silicon, 50 per cent.
- 1,000 tons ferro-silicon, 30 per cent.
- 2,000 tons ferro-chromium.
- 800 to 900 tons ferro-tungsten.
- About 50 tons ferro-molybdenum.
- 5 to 10 tons ferro-vanadium.

The total value of the alloys sold being at the present time:



FIG. 2.—FERRO-ALLOY WORKS AT UGINE.

equivalent to over 9,000,000 francs per annum. These figures alone convey some idea of the magnitude of the undertaking.

As is well known, the Société specializes largely in high-grade alloys of low carbon content, some typical analyses of these products are given below. These alloys find a ready sale, not only in France, where the development of "ternary" and "quaternary" steels has had an earlier development than in most other countries, but also in Germany, Austria and England, and even in America amongst the leading steel firms.

Of special interest are the ferro-tungsten and ferro-molyb-

denum, which account largely for the high value of the production of the works. The ferro-tungsten can be classed in two grades. The one containing about 85 per cent W, and a maximum of 0.5 per cent carbon, is chiefly employed in the manufacture of crucible tool steels. The other quality, containing 60-70 per cent W and 2-3 per cent C is largely used for the manufacture by the open-hearth process of steels containing less than 2.5 per cent W, which are used for the manufacture of springs, etc.

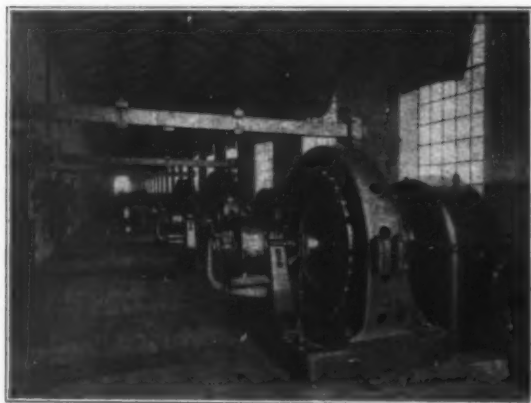


FIG. 3.—POWER HOUSE OF THE UGINE WORKS.

In both cases considerable advantage and economy are claimed over the tungsten powder which is still largely used in steel making. For assuring the most satisfactory conditions of manufacture the company has recently acquired extensive wolfram mines at Guarda, in Portugal and Puy-les-Vignes, in the province of Haute-Vienne.

#### ANALYSIS OF TYPICAL PRODUCTS.

##### *Ferro-Chromium.*

Cr .....	67.20	64.17	67.05	65.90
Fe .....	31.35	32.47	27.05	23.44
C .....	0.90	2.34	4.25	8.58
Si .....	0.19	0.38	0.60	1.26
Mn .....	0.12	0.21	0.46	0.44
Al .....	0.00	0.13	0.22	0.18
Mg .....	0.19	0.23	0.31	0.14
S .....	0.006	0.023	0.02	0.02
P .....	0.021	0.02	0.02	0.02

##### *Ferro-Tungsten.*

W .....	85.15	71.80
Fe .....	14.12	24.35
C .....	0.45	2.58
Si .....	0.13	0.36
Mn .....	0.085	0.78
P .....	0.018	0.008
S .....	0.021	0.02

##### *Ferro-Molybdenum.*

Mo .....	79.15	83.80
Fe .....	17.52	12.72
C .....	3.24	3.27
S .....	0.021	0.02
P .....	0.028	0.027

##### *Ferro-Vanadium.*

Vd .....	52.80	34.10
Fe .....	45.84	64.22
C .....	1.04	1.42
Si .....	0.09	0.12
Al .....	0.00	0.00
S .....	0.025	0.03
P .....	0.02	0.009

#### THE GIROD STEEL FURNACE.

The ingenious and interesting crucible furnace devised by M. Girod and already described in this journal (Vol. II, p. 309), has been abandoned so far as its application to the manufacture of steel is concerned. On the other hand, a furnace has been constructed and submitted to the test of extensive use which seems destined to play a not unimportant part in the inevitable extension of manufacture of steel by electrical heating processes.

This furnace is of the arc type and is characterized by the fact that it employs a single vertical electrode. The arc is maintained between this electrode and the slag which covers the metal contained in the hearth of the furnace. The molten metal is connected to the other pole of the dynamo by means of a series of water-cooled steel bars sunk in the refractory lining of the base of the hearth.

The process of treatment is in general similar to that adopted in the Héroult furnace; the superposed layers of slag being of such a composition as is required for the refining or carburization of the metal, and being renewed from time to time to assist in these processes.

The photograph of the furnace, shown in Fig. 4, does not represent it in its most recent modification, the cover or dome being now of an improved type which ensures a more complete enclosure of the furnace. The power in the present case is 300 to 350 hp., and the charge is from 1.2 to 1.5 tons.

Extensive trials have already been effected, notably by one of the leading French steel works, which has been utilizing the equipment at Ugine for the manufacture of a special steel for the construction of small pieces of armor plate, etc.

So far as can be seen the power expenditure is quite favorably low; and the possibility of eliminating the injurious constituents from the low-grade steel charged into the furnace offers no more difficulty than is the case with those furnaces which employ two electrodes arcing into the slag.



• FIG. 4.—THE GIROD STEEL FURNACE.

In the simplicity of the design and in the ease with which automatic regulation can be effected, the present furnace is certainly of considerable interest.

In the annex of the works in which this steel plant has been erected, M. Girod is also employing one of his resistance furnaces as a re-heating furnace for small steel ingots. The heating chamber of one of his multiple crucible furnaces contains the ingots to be heated, the chamber itself being surrounded by the resistance material, consisting of ferro-silicon and carbon. This type of reheating furnace is certainly worthy of consideration, for in few applications of fuel heating is the waste of coal so enormous as in the old-fashioned reheating furnaces still so largely employed in the steel industry.



## Metallurgical Calculations.

By J. W. RICHARDS, PH. D.

Professor of Metallurgy in Lehigh University.

## THE TEMPERATURE INCREMENT IN THE BESSEMER CONVERTER.

In our preceding paper (this journal, December, 1906) we have studied the generation of heat in the Bessemer converter, and its distribution. We saw in that analysis that in a typical operation, nearly one-half of the heat generated during the blow is carried out by the hot gases, about half is represented in the increased temperature of the contents of the converter, while only about 5 per cent is lost by radiation, etc. In the present paper we wish to analyze still further this question of increased temperature, which is so vitally necessary for the proper working of the process, and to calculate the relative efficiency of the various substances oxidized in causing this rise of temperature.

While at no time in the Bessemer operation is only one substance being oxidized, yet we can get the best basis for our computations by assuming a charge in operation and only one substance oxidized at a time. Whatever substance is in question, let us assume one kilogram burnt in a given short period of time, generating the heat of its combustion. With the bath at a given temperature, the air, at say 100° C., comes in contact with it, bearing the necessary oxygen. If nothing was oxidized, the oxygen and nitrogen would simply be heated to the temperature of the bath, and pass on and out, while the bath would be meanwhile losing heat also by radiation. It is evident then, that unless at least as much heat as the sum of these two items is generated, the bath will cool, and that only the excess of heat above this requirement is available for increasing the temperature of the bath and resulting gases. The proper procedure for us will therefore be to calculate, in each case, the chilling effect of the air entering, subtract this from the heat generated, and the residue is net heat available for raising the temperature of the contents of the converter and the gases, and supplying radiation losses. The latter are proportional to the time, and therefore to the amount of air used, assuming blast constant.

## SILICON.

This is burnt out in the first part of the process, during which the temperature begins low and ends high. We will therefore assume two temperatures, and calculate the thermal increment at each. We will take 1250° and 1600°. The net heat is absorbed by the bath, slag and nitrogen.

Oxygen necessary to burn one kilogram of silicon:

$1 \times 32/28 = 1.143$ kg.	
Nitrogen accompanying this oxygen	= 3.810 "
Weight of air needed	= 4.953 "
Volume of air = $4.953 \div 1.293$	= 3.831 m <sup>3</sup>
Specific heat, 100° to 1250°, per m <sup>3</sup>	= 0.3395
Specific heat, 100° to 1600°, per m <sup>3</sup>	= 0.3489
Chilling effect of air at 100°, bath at 1250°	
$= 3.831 \times 0.3395 \times 1150$	= 1496 Cal.
Chilling effect of air at 100°, bath at 1600°	
$= 3.831 \times 0.3489 \times 1500$	= 2043 Cal.
Heat generated per kilogram of silicon	= 7000 Cal.

This is, however, for cold oxygen and cold silicon burning to cold solid silica. Under the conditions prevailing we have melted silicon at the temperature of the bath, oxidized by hot oxygen to hot silica, giving a slightly different heat of combination, calculated as follows:

Heat in melted silicon at 1250°	= 480 Cal.
" oxygen required, at 1250°	= 334 "
" silica, at 1250°	= 750 "
Heat of oxidation at 1250°	
$= 7,000 + 480 + 334 - 750$	= 7,064 "

The difference from 7,000 is so small as to be within the possible error of the 7,000 itself, and we can therefore make the calculations with all the accuracy they allow, taking the ordinary heats of oxidation from the tables.

One factor of heat generation has, however, not been mentioned, viz.: the heat of combination of silica with oxides of iron and manganese to form slag. This is 148 Calories per kg. of silica when forming iron silicate, and 90 when forming manganese silicate, which quantities would become 317 and 193 Calories respectively when calculated per kg. of silicon oxidizing. The question arises, however, whether it is fair to credit all this to the silica, because this generation of heat by slag formation is really a mutual affair, chargeable to the credit of both silica and the other oxides; we should, therefore, not charge it all up to the credit of silica formation, and we do not know what part to charge to the credit of silica if we do not charge it all. In this dilemma it may be well to remember that silicon is probably oxidized before the iron and manganese, and that the heat of formation of the slag is therefore more properly considered as being generated afterwards, and therefore may be practically credited entirely to the oxidation of iron and manganese.

Resumé for oxidation of 1 kilo. of silicon:

	Cal.
Heat generated	= 7,000
Chilling effect of the blast, 100° to 1250°	= 1,496
Chilling effect of the blast, 100° to 1600°	= 2,043
Available heat, bath at 1250°	= 5,504
Available heat, bath at 1600°	= 4,957

If we assume a radiation loss proportional to the length of the blow, i. e., proportional to the air blown in, we can find out from average blows that this amounts to about 50 Calories per cubic meter of blast used. The radiation loss during combustion of 1 kilogram of silicon would therefore be

	Cal.
$3.831 \times 50$	= 192
Leaving net available heat at 1250°	= 5,312
At 1600°	= 4,765

The above quantity of heat is expended in raising the temperature of 99 kg. of bath, 2.143 kg. of silica, and 3.810 kg. of nitrogen gas, from their initial temperature. At the temperatures of 1250° and 1600°, respectively, the heat capacity of these products of the operation will be, per 1° C. rise:

Products.	Specific Heat		Heat Capacity	
	at 1250°	at 1600°	at 1250°	at 1600°
Bath, 99 kg. ....	0.25	0.25	24.8	24.8
SiO <sub>2</sub> , 2.14 kg. ....	0.37	0.43	0.8	0.9
N <sub>2</sub> , 3.02 m <sup>3</sup> . ....	0.37	0.39	1.1	1.2
Totals.	26.7		26.7	26.9

Theoretical rise of temperature:

$5,312 \div 26.7$	= 199° (bath at 1250°)
$4,765 \div 26.9$	= 177° (bath at 1600°)

Average rise, per average	
1% of silicon	= 188° C.

## MANGANESE.

As high as 4 per cent of manganese may be oxidized during the blow, and therefore this heat of combustion is sometimes important. We will calculate the net heat available for raising temperature and the rise of temperature per 1 per cent of manganese oxidized, i. e., for 1 kilo. of manganese per 100 kilos. of bath.

Oxygen necessary $1 \times 16/55$	= 0.291 kg.
Nitrogen accompanying this	= 0.970 "
Weight of air used	= 1.261 "
Volume of air used	= 0.975 m <sup>3</sup>
Chilling effect of air at 100° on bath at 1250°	
$= 0.975 \times 0.3395 \times 1150$	= 381 Cal.

Heat generated per kg. of manganese	= 1653 "
Heat of formation of $\text{MnO} \cdot \text{SiO}_2$	
1.291 kg. $\text{MnO} \times 76$	= 98 "
Total heat developed	= 1751 "
Heat available = $1751 - 381$	= 1370 "
Radiation loss = $0.975 \times 50$	= 49 "
Net available heat	= 1321 "
Heat capacity of 99 kg. of bath per $1^\circ$	= 24.8 "
Heat capacity of 2.4 kg. of slag	= 0.7 "
Heat capacity of 0.8 m <sup>3</sup> of nitrogen	= 0.3 "
Heat capacity of products per $1^\circ$	= 25.8 "

Theoretical rise of temperature:

$$1321 \div 25.8 = 51^\circ \text{ C.}$$

This is, in round numbers, one-fourth the efficiency of silicon.

#### IRON.

While it is not desired to oxidize iron, and while it is relatively less oxidizable than silicon or manganese, yet some is always oxidized because of the great excess of iron present in the converter. The amount of iron thus lost is variable, and some of it, towards the end of the blow, may be oxidized to  $\text{Fe}_2\text{O}_3$  instead of  $\text{FeO}$ , the larger part, however, oxidizes to  $\text{FeO}$ . We will make the calculations for both oxides, per kilogram of iron.

##### Formation of $\text{FeO}$ .

Oxygen necessary $1 \times 16/56$	= 0.286 kg.
Nitrogen accompanying this	= 0.953 "
Weight of air used	= 1.239 "
Volume of air used	= 0.958 "
Chilling effect of air at $100^\circ$ on bath at $1250^\circ = 0.958 \times 0.3395 \times 1150$	= 374 Cal.
Chilling effect of air at $100^\circ$ on bath at $1600^\circ = 0.958 \times 0.3489 \times 1500$	= 501 "
Heat generated per kg. of iron	= 1,173 "
Heat of formation of $\text{FeO} \cdot \text{SiO}_2 = 1.286 \text{ kg. FeO} \times 124$	= 159 "
Total heat developed	= 1,332 "
Net heat available at $1250^\circ = 1332 - 374$	= 958 "
Net heat available at $1600^\circ = 1332 - 501$	= 831 "
Radiation losses = $0.958 \times 50$	= 48 "
Net available heat at $1250^\circ$	= 910 "
Net available heat at $1600^\circ$	= 773 "
Heat capacity of 99 kg. of bath per $1^\circ$	= 24.8 "
Heat capacity of 2.4 kg. of slag per $1^\circ$	= 0.7 "
Heat capacity of 0.8 m <sup>3</sup> of nitrogen	= 0.3 "
Heat capacity of products per $1^\circ$	= 25.8 "

Theoretical rise of temperature:

$$910 \div 25.8 = 36^\circ \text{ C.}$$

$$773 \div 25.8 = 30^\circ \text{ C.}$$

This is only about one-sixth as efficient as silicon.

##### Formation of $\text{Fe}_2\text{O}_3$ .

Weight of air used	= 1.859 kg.
Volume of air used	= 1.438 m <sup>3</sup>
Chilling effect of air at $100^\circ$ on bath at $1600^\circ = 1.438 \times 0.3489 \times 1500$	= 753 Cal.
Total heat developed by oxidation	= 1746 "
Heat of formation of slag	= 159 "
Total heat developed	= 1905 "
Heat available = $1905 - 753$	= 1152 "
Radiation losses = $1.438 \times 50$	= 72 "
Net heat available	= 1080 "
Heat capacity of products	= 26 "

Theoretical rise of temperature:

$$1080 \div 26 = 42^\circ \text{ C.}$$

#### TITANIUM.

While titanium is an unusual constituent of pig iron, yet it is conceivable that titaniferous pig iron might be made and blown to steel. If so, the following calculation, based on an estimated

but quite probable value for the heat of oxidation of titanium, will show that the titanium is a valuable heat producing substance, being, in fact, weight for weight three-fourths as efficient as silicon.

Oxygen needed $1 \times 32/48$	= 0.667 kg.
Nitrogen accompanying this	= 2.222 "
Air used	= 2.888 "
Volume of air needed	= 2.250 m <sup>3</sup>
Chilling effect of air at $100^\circ$ on bath at $1250^\circ = 2.250 \times 0.3395 \times 1150$	= 878 Cal.
Heat generated per kg. of $\text{Ti}$ (probably)	= 5000 "
Heat of formation of slag—unknown.	
Net heat available = $5000 - 878$	= 4122 "
Deduct for radiation losses	= 3978 "
Heat capacity of products per $1^\circ \text{ C.}$	= 26.5 "

Theoretical rise of temperature:

$$3978 \div 26.5 = 150^\circ \text{ C.}$$

#### ALUMINIUM.

This metal is also rarely found in pig iron, yet when present it would be a powerful heat producer, as the following calculations show:

Oxygen needed $1 \times 48/54$	= 0.889 kg.
Nitrogen	= 2.963 "
Air	= 3.852 "
Volume of air	= 2.964 m <sup>3</sup>
Chilling effect of air at $100^\circ$ on bath at $1250^\circ = 2.964 \times 0.3395 \times 1150$	= 1157 Cal.
Heat generated per kg. of Al	= 7272 "
Heat of formation of slag—uncertain.	
Heat available = $7272 - 1157$	= 6115 "
Deducting for radiation losses	= 5967 "
Caloric capacity of products per $1^\circ \text{ C.}$	= 26.6 "

Theoretical rise of temperature:

$$5967 \div 26.6 = 224^\circ \text{ C.}$$

If a blow was running cold, and ferro-silicon was not on hand to add in order to increase its temperature, ferro-aluminium, or aluminium itself, would be a good substitute in the emergency.

#### NICKEL.

It is hardly probable that nickeliferous pig iron would be blown to steel, because of the waste of valuable nickel in the slag; yet if 1 per cent of nickel were thus oxidized, calculations similar to the preceding would show a net rise in temperature of the contents of the bath of about  $33^\circ \text{ C.}$

#### CHROMIUM.

Quite recently some chromiferous pig iron has been blown in the Bessemer converter, and those in charge were hampered by the lack of data as to how chromium would behave during the blow and its heat value to the converter. Technical literature has not yet been enriched by the account of the experiences, and no thermo-chemist has, as yet, determined the heat of oxidation of chromium or of formation of chromium slag. From what we know of the chemical reactions of chromium, however, it is likely that its heat of oxidation is somewhere about 3,000 Calories per kilogram. Using this, and neglecting the heat of formation of slag, we have the following approximation to its heating efficiency, assuming it to be oxidized when the bath is near to its maximum temperature:

Oxygen needed $1 \times 48/104$	= 0.462 kg.
Nitrogen entering	= 1.540 "
Air used	= 2.002 "
Volume of air	= 1.548 m <sup>3</sup>
Chilling effect of air at $100^\circ$ on bath at $1600^\circ = 1.548 \times 0.3489 \times 1500$	= 810 Cal.
Heat of oxidation (assumed)	= 3,000 "
Net heat = $3000 - 810$	= 2,190 "
Deducting for radiation losses	= 2,113 "
Caloric capacity of products per $1^\circ$	= 26.1 "

Theoretical rise of temperature:

$$2,113 \div 26.1 = 81^{\circ} \text{C.}$$

#### CARBON.

This element commences to be oxidized in large amount only towards the middle of the blow, when the temperature of the bath is high, because of the previous oxidation of silicon. It will be about right, therefore, to estimate the bath at an average temperature of  $1600^{\circ}$  during the elimination of carbon. The product is mostly CO, but partly  $\text{CO}^2$ . We will, therefore, calculate for each of these possible products separately. The net heat available, after allowing for average radiation losses, is used to increase the temperature of the products, i. e., of the bath, the nitrogen and the CO or  $\text{CO}^2$ .

#### Oxidation to $\text{CO}^2$ .

Oxygen required = $1 \times 32/12$	= 2.667 kg.
Nitrogen accompanying	= 8.889 "
Air used	= 11.556 "
Volume of air	= 8.937 $\text{m}^3$
Chilling effect of air at $100^{\circ}$ on bath at $1250^{\circ} = 8,937 \times 0.3395 \times 1150$	= 3,489 Cal.
Heat of oxidation	= 8,100 "
Heat available = $8,100 - 3,489$	= 4,611 "
Radiation losses = $8,937 \times 50$	= 447 "
Net heat available	= 4,164 "
Heat capacity of 99 kg. bath = $99 \times 0.25$	= 24.80 "
Heat capacity of $7.05 \text{ m}^3 \text{ N}^2 = 7.05 \times 0.37$	= 2.61 "
Heat capacity of $1.90 \text{ m}^3 \text{ CO}^2 = 1.90 \times 0.88$	= 1.70 "
Heat capacity of products, per $1^{\circ} \text{C.}$	= 29.11 "

Theoretical rise of temperature:

$$4,164 \div 29.11 = 143^{\circ} \text{C.}$$

Since carbon burns to  $\text{CO}^2$  principally at the beginning of the blow, while the bath is cold, we see that carbon thus consumed is about three-quarters as efficient as an equal weight of silicon in raising the temperature of the bath.

#### Oxidation to CO.

Oxygen needed $1 \times 16/12$	= 1.333 kg.
Nitrogen accompanying	= 4.444 "
Air used	= 5.777 "
Volume of air	= 4.469 $\text{m}^3$
Chilling effect of air at $100^{\circ}$ on bath at $1600^{\circ} = 4,469 \times 0.3489 \times 1500$	= 2,339 Cal.
Heat of oxidation	= 2,430 "
Heat available = $2,430 - 2,339$	= 91 "
Radiation losses = $4,469 \times 50$	= 233 "
Net heat available = $91 - 233$	= -142 "
Heat capacity of 99 kg. of bath = $99 \times 0.25$	= 24.8 "
Heat capacity of $3.5 \text{ m}^3 \text{ of N}^2$	} $5.4 \times 0.39 = 2.1$ "
Heat capacity of $1.9 \text{ m}^3 \text{ of CO}$	
Heat capacity of products	= 26.9 "

Theoretical rise of temperature:

$$-142 \div 26.9 = -5^{\circ} \text{C.}$$

The result is, therefore, that when carbon burns, as it mostly does, to CO, and the temperature of the bath is high, there is practically no further rise of temperature, for the heat of oxidation is barely sufficient to counteract the chilling effect of the air and to supply radiation and conduction losses.

In the above calculations, no allowance was made for heat required to separate carbon from its combination with iron, or for the variation in the heat of combination of carbon with oxygen from the combination heats at ordinary temperatures. The former is not known, or perhaps is very nearly zero. The heat of oxidation of liquid carbon at  $1250^{\circ}$  to  $\text{CO}^2$  or at  $1600^{\circ}$  to CO is calculated as follows:

Oxidation 1 kg. C to $\text{CO}^2$ at $0^{\circ}$	= 8,100 Cal.
Heat to raise 1 kg. C to $1250^{\circ}$	= 505 Cal.
Heat to liquefy 1 kg. at $1250^{\circ}$	= 129 "
Heat to raise $2.67 \text{ kg. O}^2$ to $1250^{\circ}$	= 779 "

Heat to raise reacting substances to $1250^{\circ}$	= 1,413 "
Heat in 3.67 kg. $\text{CO}^2$ at $1250^{\circ}$	= 1,493 "
Heat of reaction at $1250^{\circ}$ ( $8,100 + 1,413 - 1,493$ )	= 8,020 "

For production of CO, at  $1600^{\circ}$ , the correction is larger, as is seen from the following:

Oxidation 1 kg. C to CO at $0^{\circ}$	= 2,430 Cal.
Heat to raise 1 kg. to $1600^{\circ}$	= 680 Cal.
Heat to liquefy 1 kg. C at $1600^{\circ}$	= 156 "
Heat to raise 1.33 kg. $\text{O}^2$ to $1600^{\circ}$	= 554 "

Heat to raise reacting substances to $1600^{\circ}$	= 1,390 "
Heat in 2.33 kg. of CO at $1600^{\circ}$	= 1,104 "
Heat of reaction at $1600^{\circ}$ ( $2,430 + 1,390 - 1,104$ )	= 2,716 "

The use of this corrected value makes the oxidation of C to CO give a small net heat development, with consequent slight rise of temperature, instead of the slight cooling effect before calculated. The conditions are so nearly even, however, that a slight increase of temperature or slowing up of the blow would wipe out the heat excess.

#### PHOSPHORUS.

This is the last important element to be considered, and is always eliminated after the carbon, at the maximum bath temperature, which we will assume, for calculation, at  $1600^{\circ}$ . The heat generated, at ordinary temperatures, is 5892 Calories per kilogram of solid phosphorus. Per kilogram of liquid phosphorus it would be only 5 Calories more, or 5897 Calories. For the reaction at  $1600^{\circ}$  we would have a different value, probably some 500 Calories more, but the necessary data concerning the specific heats of P and  $\text{P}^2\text{O}^5$  are not known, and we must omit this calculation. The heat of combination of iron and phosphorus is also a doubtful quantity. Ponthière places it as high as 1,397 Calories per kilogram of phosphorus, but this appears altogether improbable, since another experimenter could obtain no heat of combination at all. As concluded in another place, I advise for the present omitting this questionable quantity.

The phosphorus pent-oxide forms  $3\text{CaO} \cdot \text{P}^2\text{O}^5$  with the lime added, but since there is always more lime present than corresponds to these proportions ( $3\text{CaO} : \text{P}^2\text{O}^5 :: 168 : 142$ ), the calculation of heat of formation of the slag must be based on the amount of  $\text{P}^2\text{O}^5$  formed (1123 Calories per kilogram of  $\text{P}^2\text{O}^5$ ). This amounts to a considerable item. On the other hand, the lime needed for slag is put in, usually preheated, for the sole purpose of combining with the  $\text{P}^2\text{O}^5$ . It seems, therefore, only right to charge the phosphorus with the heat required to raise this lime to the temperature of the bath. The lime added averages three times the weight of  $\text{P}^2\text{O}^5$  formed, and is preheated usually to about  $600^{\circ}$ . Assuming these conditions, the following calculations can be made per kilogram of phosphorus oxidized:

Oxygen required	= 1.29 kg.
Nitrogen accompanying	= 4.30 "
Air used	= 5.59 "
Volume of air	= 4.32 $\text{m}^3$
Heat of formation of slag, $2.29 \text{ kg. P}^2\text{O}^5 \times 1123$	= 2572 Cal.
Heat of oxidation of phosphorus	= 5897 "
Total heat developed	= 8469 "
Chilling effect of air at $100^{\circ}$ on bath at $1600^{\circ} = 4.32 \times 0.3489 \times 1500$	= 2261 "
Chilling effect of lime ( $600^{\circ}$ to $1600^{\circ}$ ) ( $2.29 \times 3$ ) $\times 0.328 \times 1000$	= 2253 "
Chilling effect of blast and lime	= 4514 "
Heat available = $8469 - 4514$	= 3955 "
Radiation losses = $4.32 \times 50$	= 216 "
Net heat available	= 3739 "
Heat capacity 99 kg. of bath = $99 \times 0.25$	= 24.8 "
Heat capacity $3.4 \text{ m}^3$ of $\text{N}^2 = 3.4 \times 0.39$	= 1.3 "
Heat capacity 6.9 kg. of slag = $6.9 \times 0.3$	= 2.1 "
Heat capacity of products, per $1^{\circ}$	= 28.2 "



Theoretical rise of temperature:

$$3739 \div 28.2 = 133^{\circ} \text{ C.}$$

If the lime were added cold, its cooling effect would be 883 Calories greater, the net heat available would be 883 Calories less, and the calculated rise of temperature  $31^{\circ}$  less, or  $102^{\circ} \text{ C.}$  Using the preheated lime we can regard phosphorus as being practically two-thirds as efficient, weight for weight, as silicon; with cold lime, about one-half as efficient.

#### RESUME.

Heat effect of oxidizing 1 kilogram of element.

	Heat of Oxidation.	Formation of Slag.	Total Heat Developed.	Chilling Effect of Blast, Radiation, etc.	Net Heat Available for Raising Temperature.	Theoretical Rise of Temperature.
Silicon.....	7,000	.....	7,000	1,688	5,312	188°
Manganese.....	1,653	98	1,751	430	1,321	51°
Iron (to FeO).....	1,173	159	1,332	422	910	33°
Iron (to Fe <sub>2</sub> O <sub>3</sub> ).....	1,740	159	1,905	825	1,080	42°
Titanium.....	5,000	.....	5,000	1,022	3,978	150°
Aluminium.....	7,272	.....	7,272	1,305	5,967	224°
Nickel.....	1,051	159	1,210	378	832	33°
Chromium.....	3,000	.....	3,000	887	2,113	81°
Carbon (to CO <sub>2</sub> ).....	8,100	.....	8,100	3,936	4,164	143°
Carbon (to CO).....	2,430	.....	2,430	2,572	-142	-5°
Phosphorus.....	5,897	2,572	8,469	2,477	3,739	133°
				2,253*		

\* Chilling effect of lime added, preheated to  $600^{\circ}$ .

It must be observed that the above table is for comparison only; it cannot be used for an actual case, such as when 1 per cent of silicon, 3 of iron, 4 of carbon and 2 of phosphorus are oxidized. In such a case, the rise in temperature would be only very roughly:

From silicon.....	$1 \times 188 = 188^{\circ}$
From iron.....	$3 \times 33 = 99^{\circ}$
From carbon.....	say $= 0^{\circ}$
From phosphorus.....	$2 \times 133 = 266^{\circ}$

$$\text{Total} = 553^{\circ} \text{ C.}$$

It is to be recommended that in each specific case the calculation be made for the specific conditions obtaining, such as temperature of the metal at starting, temperature of the blast, time of the blow (as far as this affects radiation and conduction losses), proportion of carbon burned to CO, free oxygen in the gases, moisture in the blast, temperature and quantity of lime added, corrosion of lining. When all these items and conditions are taken into account there will be room for only small discrepancy between the calculated and the observed rise of temperature. The chief items needing experimental research at present are: The specific heat of the melted bath, the specific heat of the slag, the heat of combination of various elements comprising the bath, the heat of formation of the slag, and the heat of oxidation of some of the rarer elements, such as titanium and chromium. Such establishments as the Carnegie Institution could not do the cause of metallurgy better service than to subsidize metallurgical laboratories for the determination of such data.

**Western Association of Technical Chemists and Metallurgists.**—The second general meeting of this Association was held in Salt Lake City, Utah, from Dec. 27 to 29, 1906. The program contains the following papers: F. W. Traphagen, on the smelter smoke problem; F. A. Thomson, on a sampling experience with metallics; S. Rickard, on an electric method of preventing boiler scale; F. Leonard, on cyaniding practice on Boulder County ores; H. C. Parmelee, on the determination of insoluble silicious residue. Numerous visits and excursions to metallurgical plants were also on the program.

#### Modern Gold Milling.

By GEORGE P. SCHOLL, PH. D.

Within recent years there have been a great many changes in the construction of mill buildings, and the progress made in the structural and engineering features is quite remarkable. The modern mill already in its exterior appearance clearly shows the change which has taken place. Instead of the ramshackle, hastily put up structures which were so much in evidence during former years, and which showed plainly that they were only put up in the cheapest possible manner and with hardly any thought to permanency, we now see well-designed buildings.

In the more modern of mill buildings structural iron has replaced wood almost altogether, and in some of the modern plants the use of iron has been carried so far that even the mill bins and the elevator casings are constructed of sheet iron. Amply lighted and well ventilated and with sufficient room allowed for the inspection and accessibility of the machinery, these mills bear evidence that they were put up in a substantial manner, and with a view to secure ease of supervision.

The interior aspect of a modern mill is also altogether different from one of the older type. It does no more present the aspect of a lumber yard, it has no shaky floors and unsubstantial foundations, and no apparatus are stuck away in corners without any regard being paid to their accessibility for supervision and repair.

With its substantial cement floors, its solid mortar foundations, it shows clearly that it is intended to facilitate operations in every possible respect; and to fulfill the purpose for which it was built, namely, to treat large masses of low grade ore as cheaply as possible.

There is no doubt that in the mining industry the future does not belong to the comparatively small quantities of bonanza ore but to the large masses of low grade propositions. This change in the aspect of the mills typifies the change in the whole system of metallurgy, because these low grade propositions have to be worked along scientific lines, and every possible item of value has to be saved, so that the slipshod methods employed at the older mills, where very frequently nobody seemed to care how much gold and other values were lost in the tailings, can no longer be employed.

Modern mill practice, therefore, differs essentially from that which was prevalent even a few years ago. The writer has recently had occasion to visit a large number of modern mills, and it was extremely interesting to notice the changes which have taken place.

In the modern plants a good deal of attention is being paid to securing good and substantial foundations for the apparatus, and especially the stamp batteries. Liberal use is made of concrete for this purpose, and the foundation work is sometimes very costly, but the increase in cost is more than made up for by the solidity and freedom from jarring and vibration of the building. In regard to the battery foundations, modern tendencies are overwhelmingly in favor of the concrete foundation. The prejudice which unquestionably existed in the mind of many managers, even men of wide experience in stamp mill practice, against the concrete foundations, seems to be fast disappearing, although it is by no means altogether overcome.

It is quite true that the opponents of the concrete foundation could point to instances where the foundations had not given satisfaction, but the greater part of the trouble was undoubtedly caused by the unfamiliarity of the men who built the foundation, with the kind of material required for this work, and perhaps their unfamiliarity with concrete work in general. It is not always easy in far-away mining regions to secure men particularly fitted for carrying out a delicate job of work which requires care and attention, as concrete work undoubtedly does, and the tendencies to carelessness in the working of the material and the proper mixing and proportioning of the ingredients are

always present. Moreover, a good deal of care is required in getting the battery mortars to bear evenly with their lower surface on the top of the concrete foundation, and any neglect in this regard is apt to lead to disastrous results.

The interposition of a sheet of rubber to cushion the blow is, of course, an additional safeguard in this respect, but it is much more advisable to pay careful attention to the leveling of the top of the foundation and to use a comparatively thin sheet of rubber than to rely upon a thicker intervening layer. A well-built concrete foundation is sure to last indefinitely, and to cause much less trouble than a wooden foundation, which latter is undoubtedly the weakest point in the battery, and a source of considerable trouble when it needs repairs and renewals.

A point to which particular attention should be paid in the construction of concrete foundations is that the proper amount of cement should be allowed in the mixture. This is one of the instances where it does not pay to save on the cement and use too high a proportion of inert materials. It should always be remembered that the cement is the only material that holds the concrete together and gives strength to the mixture, and that a structure which is so much subjected to vibration and strains as a battery foundation is needs to have considerable strength.

The tendency of course to cheapen the cost of the concrete by decreasing the amount of cement is always present, especially when the work is done by independent contractors. One instance of such a procedure came to the notice of the writer, where the battery foundation had cracked on account of the poor material used by the contractor and had to be largely replaced. In this instance the contractor had deliberately and by collusion with an employee who was supposed to watch the work, disregarded the proportions specified in the contract, and had used a much smaller proportion of cement. Such occurrences are, however, not so exceedingly rare as might be imagined, and probably a good deal of the unsatisfactory behavior of some concrete foundations may be traced to a similar cause, perhaps not to open dishonesty as in the example mentioned, but to ignorance.

Heavy anvil blocks interposed between the mortar and the concrete foundation are unnecessary and objectionable. It is sufficient if the mortar be provided with a rather wide base, and that the underside be machined off carefully, so as to be absolutely straight and have an even bearing on the concrete foundation.

As far as the batteries themselves are concerned, modern tendencies seem to lean towards an increased weight of the stamps, except, of course, in such cases where the character of the ore is such that it slimes easily, and an excessive sliming is objectionable. Stamps of 1,350 pounds weight are by no means a rarity, and good results have even been reported from South Africa with still heavier stamps, of 1,450 pounds weight.

This practice is in line with the general increasing tendency to regard the stamps simply as one link in the cycle of crushing machinery, to be used for what might be called preliminary crushing, especially when they are supplemented by tube mills. In such cases the aim is, of course, directed towards putting through the batteries as large a quantity of material as possible, and to increase the aperture of the battery screen. We thus hear from South Africa of such stamp duties as 7.8 tons per stamp per day, in the Knights Deep Mill, the latter figure having been obtained with 1,450-pound stamps.

One feature of a modern mill, and which is much to be recommended, is the adoption of suspended ore feeders to the batteries, with the attendant advantage of every accessibility of the feeders as well as the stamp batteries for the purpose of inspection and repair. The unobstructed floor space thus gained behind the batteries is of great assistance in facilitating the supervision of the machinery and providing the necessary accessibility to the parts.

Automatic ore feeders have come to be recognized as such essential auxiliaries in stamp mill that there is no need on dwelling upon them at this place. There are a number of types

in use, and others are still being introduced, the aim of the constructors being to improve and facilitate the operation of the feeding devices and to provide for the possibility of easy regulation.

It was also clearly evident that a great deal of attention was paid to the fine grinding of materials and the regrinding of the tailings. For this purpose the tube mill has now established a well-defined place for itself in metallurgy, and continues to make its way. As far as the tube mills are concerned, it is not to be denied that they are certainly very effective instruments for fine grinding.

There are already a good many data relating to the subject of tube milling, but as yet no method has been elaborated by which the work done by the various tube mills can be strictly compared. The common method at present in use simply relies upon the diminution of a certain size, usually the product which passes through a 60-mesh sieve, and the difference between the percentage remaining on the sieve before and after the passage of the material through the tube mill is taken to indicate the amount of crushing work performed by the mill.

This method is not altogether satisfactory, inasmuch as it does not take account of the difference in the battery screens, the pulp of which feeds the mills, and, therefore, results obtained at different plants are not strictly comparable. It has lately been proposed to calculate the relative surface area of the particles of crushed material before and after its passage through the tube mill, and thus to ascertain the efficiency of the crushing work done by the latter. Some such method will have to be adopted in order to be able to strictly compare results.

The introduction of the tube mill has been very rapid, especially in Western Australia and in the Transvaal, and the mills have given upon the whole very good satisfaction. It seems to have been clearly established that short mills are more effective than longer ones, and it is stated that, for instance, in Australia, the majority of the mills are 13 feet long, and have been found to do more effective work per horse-power than the others. This experience has also been confirmed at the El Oro (Mexico) plant.

As far as the lining of the tube mills is concerned, it appears that siliceous is at the present time the material which is mostly in use, with the possible exception of Australia. The siliceous lining has shown itself to be much preferable to the chilled iron or manganese steel linings, especially in those cases where the material going into the tube mill is ground fine in one passage through the mill.

On the other hand, in Australia, chilled iron liners are given the preference, but in those mills the material is constantly returned to the mill until it passes 150-mesh screen, so that the material with which the mill deals is in a much finer state of division from the start.

In the United States, siliceous liners are employed in the majority of cases, and it has been found that a better wear is obtained by putting the bricks which compose the lining on edge instead of laying them flat. The bricks generally are 2½ inches thick, 4 inches wide and 8 inches long.

The subject of tube milling is gradually beginning to be understood, and the best conditions for the cheapest and most effective operation are being ascertained. It has been found among other things that a great deal depends upon the previous classification of the material that is to be ground in the mill.

In the beginning, the lining as well as the pebbles were subjected to a great deal of wear and tear, by reason that the mill was fed with pulp of too thin a consistency, or did not receive enough of the material. It has been ascertained that the tube mill will do its best work when it receives the pulp in a rather thick condition, say, about 50 per cent of moisture, and that under these conditions the output will be a maximum and the wear and tear of liners and pebbles a minimum.

In order to get this favorable consistency of the pulp it becomes necessary to introduce a classifier into which the pulp

enters before it is fed into the mill. It has now become recognized that such a classifier is a very important auxiliary, and that a sufficient amount of surface should be allowed so that the settling of the pulp can proceed properly.

The more modern plants accordingly have classifiers of ample size. They thus gain the two-fold advantage of a maximum output and minimum wear of liners and pebbles as well as relieving the mills from a great amount of material which is already fine enough, and therefore need not be passed through the mill.

The question of the liners, as mentioned above, is being solved in a different fashion in the various countries where tube mills have been introduced, in accordance with the character of the ore. A notable advance in the construction of the lining has been made by Mr. P. Barry, the chief engineer of the Waihi Gold Mining Co., of New Zealand.

Mr. Barry intends to facilitate the laying and the repair of the tube mill lining, and for this purpose he constructs the lining of a series of rectangular hollow cast-iron segments, so that it presents a sort of honeycombed appearance. The segments are curved to conform to the shape of the mill. Pieces of silex are cemented into the individual segments.

It is reported that this construction has worked with great satisfaction at the Waihi mine. The construction is particularly adapted to a quick repair of the lining when the latter, as happens quite frequently, is worn out in places. It is then only necessary to replace the worn-out segment or segments by new ones, and the mill is ready to start again in short order.

The time required for the repair of the lining, which very frequently is a serious and costly item, has been decreased materially at the Waihi plant by the installation of this lining. The reduction in cost is given as about \$1,000 per mill per annum.

The old controversy between the tube mills and pans is by no means settled, and probably will not be for some time to come. It is, however, becoming customary to pass the product issuing from the tube mills again over a series of amalgamating plates, in order to catch any free gold which may have become liberated in the mill.

Such is, for instance, the practice at the Liberty Bell Mill, Telluride, Col., lately visited by the writer, where three Abbé tube mills are working, two being in operation and one in reserve. The mill has a total capacity of 80 stamps, crushing through a 14-mesh screen. The stamps weigh 850 pounds and drop 100 times per minute 6 inches high. The stamp duty is given at 4.3 tons per 24 hours. The pulp passes over the plates, and after flowing into a Dorr classifier it is delivered to the tube mills in a thickened condition. One of these classifiers handles the product of 40 stamps.

The Dorr classifier at this plant is stated to have given good service, although its settling capacity is somewhat too small in this particular instance. The classifier is a comparatively simple apparatus, inasmuch as it consists essentially of a longitudinal trough with inclined sides, which is fitted with a sort of mechanical scraper. This scraper is constructed with a series of rakes, and works in such a manner that it constantly scrapes up the heavy material which has settled in the bottom of the tank, and moves it over the inclined bottom until it reaches the feed arrangement of the mill.

The latter are 22 feet long and 5 feet wide, and are supported by means of two tires, one near each end, upon four small wheels or friction rollers. The tube mills are lined with silex bricks 4 inches thick set on edge, and are run at about 25 r. p. m.

The spiral feed of the Abbé type of mill is stated to have been very satisfactory. The tube mills grind about 95 tons each per 24 hours, and about 85 to 90 per cent of the material which they discharge will pass through a 100-mesh screen.

The pebbles used in the mill are the regular imported product, of an average size of  $2\frac{1}{2}$  inches, and their consumption is given as 1.3 pounds per ton of ore ground.

The mill carries a load of pebbles of about 14 tons, being two-thirds filled. The product discharged from the mill has the following fineness: On 40-mesh, 40 per cent; on 80-mesh, 13 per cent; on 100-mesh, 15 per cent; on 200-mesh, 23 per cent; through 200-mesh, 48 per cent.

The pulp issuing from the mills, as mentioned above, passes again over a system of amalgamating tables, six plates being provided for each mill. From there it goes to five large settling tanks, 30 feet x 10 feet, preparatory to being charged into the agitating vats for cyaniding. It is an interesting fact that the whole crushing process at the Liberty Bell mill takes place in cyanide solution, but the latter is comparatively weak, as it only stands in the battery mortars at about 1 pound of cyanide per ton. It is strengthened up in the agitators to  $1\frac{3}{4}$  pounds of cyanide per ton.

The question of crushing with or without cyanide solution has been discussed for some time, and it is generally conceded that under ordinary circumstances the practice is not to be recommended. On the whole, it is not so much a metallurgical as a commercial question.

If the attack of the cyanide solution on the plates and the deterioration of the latter is sufficiently high in cost to counter-balance the amount of gold saved, or the greater percentage extracted in cyaniding, it is obviously a better policy to discard crushing in cyanide solution. At the Liberty Bell it has been decided that the advantages gained are sufficient to continue its use, especially in view of the use of the tube mills, which aid very effectively in the extraction of the values, besides serving as grinding machines.

In contradistinction to that, at both mills of the Smuggler Union Gold Mining Co., which are located only a short distance from the Liberty Bell, and work on a similar ore, the practice of crushing in cyanide solution has not been adopted. At this place the tube mills are not utilized in the ore treatment.

The tube mill has also, but comparatively recently, been introduced as an apparatus for the regrinding of tailings, with a view to obtain a closer saving of the values. When it is used in such a capacity the object is, of course, not fine grinding preliminary to subsequent cyanidation, but simply a regrinding of the tailings obtained from the treatment of the battery pulp on the first set of coarse tables.

The tube mill in that case acts simply as an auxiliary to the stamps, and serves in the same capacity as the Huntingdon or Bryan mill or similar apparatus, which are used for the same purpose. When it is employed in such a manner, the object may simply be to obtain an improved recovery by the regrinding of the tailings, which are then subsequently treated on concentrating tables and their values recovered. Thus the values contained in the tailings and which in the older types of mill were allowed to go to waste are saved.

In such cases the battery screen may not be changed, and the mills are not used for what might be called very fine grinding or sliming, but simply for comminuting the ore sufficiently to recover the metallic contents. In cases where the gold values in the ore warrant it, auxiliary amalgamating tables may also be introduced between the tube mills and the concentrating tables, in order to save the additional amount of free gold liberated in the crushing operation in the mill.

In other cases an object in installing the tube mill may be the increase of the capacity of the plant by the use of a coarser screen in the batteries, and the utilization of the tube mill as an auxiliary crushing machine, for recrushing the coarse tailings obtained after the first coarse concentration. Whether the tube mill is the most economical instrument to be used in such cases as mentioned above, does not seem to be altogether certain. In any case final judgment cannot be pronounced until the mills have been in operation for some time and comparative data are available.

The initial cost of the tube mills is, of course, quite an item, and the necessity of keeping a spare unit of rather large dimen-



sions is not to be disregarded. Still, the plants who have adopted them for auxiliary crushing seem to be satisfied with the results, and at least one large company has recently installed a new plant for the treatment of ore similar to that dealt with in its first plant, and has adopted tube mills in the second plant also. In view of these facts the future of the tube mill as an essential adjunct in modern metallurgical practice seems assured.

The cost of siliceous linings for tube mills is quoted as \$0.6 to 1.00, according to quality, per square foot of lining area, when the bricks are laid flat, so that the lining has a thickness of  $2\frac{1}{2}$  inches. When the bricks are set on edge, making the thickness of the lining 4 inches, the cost increases to about \$0.9 to 1.60 per square foot of lining area. These prices apply to the material f. o. b. New York. A lining of 4-inch thickness might reasonably be expected under average conditions to last about eight to ten months, and probably will last much longer, judging from experience obtained in South Africa. The cost of pebbles f. o. b. New York is about \$11.00 to \$16.00 per ton, and their consumption may be calculated on an average as about 2 pounds of pebbles per ton of ore crushed. It will vary of course with the character of the ore and the fineness of the product required. As far as the cost of tube milling is concerned, a large representative machinery house, which has had much experience in the installation of tube mills, quotes the following figures as representing the practice at a representative tube mill plant: Power, 3.6c.; wages, 2.82c.; liners, 2.0c.; maintenance (including tables), 1.36c.; pebbles, 1.16c.; sundry stores, 0.48c.; a total of 11.42c. per ton. The costs at the Robinson Deep Gold Mining Co., on the Witwatersrand, are given by Mr. W. R. Dowling, the manager of the plant, in the journal of the Chemical, Metallurgical and Mining Society of South Africa, 1906, p. 308, as follows, for a total of 29,235 tons: Maintenance, 0.72c.; wages, 2.86c.; power, 3.78c.; sundry stores, 0.52c.; pebbles, 1.4c.; liners, 1.86c., or a total of 11.14c, which figure shows a remarkably close correspondence to the first quoted one.

The cost of tube milling at the Combination mill, Goldfield, Nev., with over 40 cents per ton of ore stamped represents about the maximum, on account of the high cost of labor and freight. Mr. F. L. Bosqui, in a paper read at the recent Denver meeting of the American Mining Congress, gave the following data about this plant: Cost of  $2\frac{1}{2}$  inches siliceous lining, laid in mill, \$323.50; life of lining, four months; cost of lining, 7.7 cents per ton of ore stamped; cost of pebbles delivered at Goldfield, Nev., \$71 per ton; consumption of pebbles, 2.03 pounds per ton of ore stamped; cost of pebbles, 7.1 cents per ton of ore stamped; horse-power, 25 hp. at \$4.25 per hp. per month, cost 26.7 cents per ton of ore stamped. The total cost is, therefore: Pebbles, \$0.71; lining, \$0.77; power, \$2.67 = \$4.15. The tube mill in this case is 12 x 4 feet, and handles about 24.6 tons per day, sliming the product of a Bryan mill with a 40-mesh screen.

In close connection with the increased use of tube mills for fine grinding stands the progress made in cyanidation, and especially the processes intended to facilitate the treatment of the slimes. The efforts seem to be mainly directed towards cheapening the slimes treatment. In these days of comparatively low-grade ores the reduction of costs is a very important consideration, in fact, for many ores a *conditio sine qua non*. It is sufficiently evident that the treatment of the slimes in filter presses is a very laborious and costly operation on account of the skilled labor required and the comparatively small capacity of a press. On the other hand, the decantation method of slimes treatment involves a great bulk of solution and the installation and up-keep of quite a number of tanks, and on that account it has not been regarded with a great deal of favor. The main object of the new methods of slime treatment has, therefore, been to lower the cost of filter-pressing by providing filtering units of cheap construction and very large filtering area, and to facilitate the discharge of the resulting cake of slime by adopting mechanical means for its removal.

One of the processes based upon these ideas and which has been installed at several places is the Moore filtering process. The process depends essentially on filtering the slimes pulp by the aid of a vacuum, the residual cakes of slime adhering to the canvas filter frames of special construction. When the cakes have reached a sufficient thickness, the whole filter set, comprising a series of frames, is raised from the filtering tank and introduced into a wash-water tank, where the application of the vacuum is continued until the strength of the solution has fallen down to the required amount.

When the washing has been completed, the filter set is raised again and run over a discharge hopper, the vacuum still being maintained until the excess of moisture is removed. After that air pressure is turned into the filter frames, and the cakes are gradually loosened until they drop off. This was essentially the practice at the first installation at the Standard Mill at Bodie, Cal., where the filtering arrangement consists of forty-nine filter frames, 16 feet long and 5 feet wide; one set of filters had a total filtering area of 7,840 square feet. The results at Bodie, according to R. Gilman Brown (*Min. and Scient. Press*, Vol. 93, p. 192), have been very satisfactory, and have resulted in a saving of about \$2.00 per ton. The costs are given as follows: General expenses, including superintendence, watchman, assays, insurance, taxes, chemicals, supplies, etc., \$1.202; regrinding, \$0.572; Moore process, \$0.314; zinc room, \$0.261; a total in the slimes plant of \$2.349. This cost is expected to be brought down to well below \$2.00.

The process has also recently been installed at the Liberty Bell Mill, at Telluride, Col., mentioned above, and is in active operation. As the practice at the Liberty Bell is based upon modern metallurgical ideas, a brief description will prove of interest. The tube mill practice at the plant has been mentioned above. The pulp discharged from the tube mill, after passing over the secondary amalgamating plates, is conducted to five large settling vats, passing through spitzluten on the way, where the coarse product is caught and returned to the tube mills. The settling vats are 33 feet in diameter and about 10 feet deep.

The pulp is allowed to settle for 12 hours, after which about 5 feet of the clear solution are decanted off and pass directly to the strong zinc boxes. By means of central discharge valves the pulp is then conducted to six agitating vats of the Hendryx type. They are provided with a central tube, which contains a spiral screw; the latter scoops up the pulp from the bottom and delivers it at the top of the vat, thus giving it a thorough mixing and agitation. The agitator vats are 17 feet in diameter, and they have a cone-shaped bottom, their depth being 11 feet to the top of the cone. The capacity of the vats is 35 tons of dry ore per charge. In these agitators the cyanide solution is strengthened up to  $1\frac{1}{2}$  pounds of cyanide per ton, and the pulp is agitated for about 12 hours, after which it is discharged into a storage tank by means of a centrifugal pump. Lead acetate is added to the solution in the agitating vats, and it is stated that the adoption of this practice has led to very beneficial results in the extraction, especially of the silver values. The main object in adding lead acetate is the removal of any soluble salts of lead and mercury in the form of insoluble lead and mercury sulphide, so that they cannot retard the solution of the gold and silver or reprecipitate any silver already dissolved.

From the storage vat the pulp goes to the Moore filtering plant, which is provided with six tanks, two of which are used for carrying on the filtering operation and the remaining four for the displacement of the filter cakes. The practice in the Moore plant differs in several respects from that adopted at Bodie, Cal. The assistant general manager of the Liberty Bell, Mr. E. H. Nutter, has had much practical experience in the practice of the Moore process, having been connected previously with the management of the Bodie plant.

As stated above, the length of the filters at the Bodie mill is 16 feet. This has been cut down to half of that length at the

Liberty Bell plant, and the number of filters in a set has been increased, so that one filter unit now comprises sixty-six frames, covered with 20-ounce duck, each frame being 8 feet long and 5½ feet wide. A filter unit, therefore, has a total filtering area of nearly 6,400 square feet. This shorter length of the filters facilitates the handling considerably.

The filtering set is introduced into one of the filtering tanks and the solution is drawn through the filter, agitation being kept up in order to prevent the pulp from settling. According to Mr. W. E. Tracy, the superintendent of the mill, in *Eng. and Min. Journ.*, Vol. 82, p. 149, the filter sets remain in the tank for 1 hour, and during this time about 16 tons of solution are drawn through and a cake of ¾-inch thickness is formed. The vacuum used is about 17 to 19 inches.

After this operation the basket is transferred by means of an hydraulic crane operated by an electric motor to a displacing tank; the gold-bearing cyanide solution is then displaced by water, but no intermediate weak cyanide solution is used. The cakes of slime, after they leave the filtering tank are stated to contain about 33 per cent of moisture, 75 per cent of which is drawn off in from 30 to 40 minutes, after the filter set has been transferred to the displacing tank. The strong solution from the Moore plant, together with the solution drawn from the settlers, is run to the ten strong zinc precipitation boxes, while the weak solution runs through the weak zinc boxes and then to waste. Before the strong solution goes to the boxes it passes a clarifying filter arrangement. The displacement is stated to be continued, until the discharge from the vacuum pump shows .35 pound of cyanide per ton, and at that point there are left about 5c. to 7c. of dissolved values per ton of dry slime and about 3c. of cyanide.

A further important modification from the original practice at Bodie has also been introduced, namely, the cakes are not blown off from the filter frames by means of compressed air, but by water at 10 pounds pressure under water in the displacing tank. This procedure would seem to present decided advantages, inasmuch as the strain caused by the action of the air on the filters must be quite considerable and conducive to leaks, and leaky filter frames in such a process are a source of considerable trouble. The process as carried out at the Liberty Bell mill is stated to work satisfactory.

The total time required for one cycle of operations is about 2¼ hours, comprising 1 hour for filtering, 10 minutes for transfer of the set of filters to the unloading tank and about 1 hour and 35 minutes for washing the cakes, discharging the residue and transferring the filter unit back again to the filtering tank.

A second process, which aims at reducing the cost of the filtering operation is the Cassel-Butters process, installed at the mill of the Combination Mine, Goldfields, Nev. It differs essentially from the Moore process by the fact that the filtering and removal of the filtered cakes take place in the same tank, the necessity for handling the filters being thus done away with. It is claimed that this arrangement leads to increased facility of operation and economy in labor. As installed at the small mill of the Combination Mine, the filtering apparatus, according to Mr. F. L. Bosqui (*Min. and Scient. Press*, Vol. 93, No. 16), consists of twenty-eight 5 x 10-foot frames, set 4½ inches apart in a box 10 feet square with a pointed bottom, which is inclined at an angle of 50°.

The slime pulp having been introduced at the point of the bottom, filtering is continued for about 20 minutes with a vacuum of 22 inches, and a cake of ¾ to 1 inch thickness obtained on both sides of a frame. The pulp is then drawn off into a reservoir, and a weak cyanide solution is introduced, which serves as a wash. After the cakes are thoroughly washed this solution is drawn off into a special vat and the box is then filled with water, after which water, under pressure, is admitted to the interior of the filter frames and the cake of slime thus forced off the frames. The slime drops into the pointed bottom of the box, and is removed by sluicing it out.

The whole operation is stated to require 3½ hours, 9 tons of dry slime being treated at each charge by one man. The cost of filtering has decreased to 26c. per ton, as against 96c. with an old type of filter presses.

As stated above, filter pressing in the usual types of filter presses is quite a costly operation. Mr. C. W. Merrill, however, the well-known manager of the Homestake plant, has devised a new type of press, the principal points of interest about which are its large size and the fact that the filter cakes can be discharged by sluicing without opening the press. A press of this construction, 45 feet long, has a capacity of 26 tons, and is provided with ninety-two filter frames, 4 x 6 feet. The filtered cakes are 4 inches thick. A large plant is now in course of erection, and it is expected to treat slimes of an average value of less than \$1.00 for about 25 cents. A notable departure from the usual practice is also the precipitation, at the Homestake mill, of the gold-bearing cyanide solution with zinc dust instead of zinc shavings.

In line with the above mentioned advances in the cyanide process is the installation, on a large scale, in a 100-stamp mill, of the Garvin electrocyanide process. This process, as described in *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*, Vol. 4, p. 247, is based upon the thorough agitation of the pulp to be cyanided in a very weak solution of cyanide, the operation being supplemented by the electrolysis of the gold-bearing cyanide solution. In the plant under consideration, which is now being completed at Palmer Mountain, Wash., twenty Garvin apparatus will be installed, namely, one for each five stamps. The latter have a weight of 1,050 pounds, and drop 6 inches 105 times per minute. The battery screen is 40-mesh. As the ore is not a very hard one to crush a stamp duty of about 4 tons is expected. The Garvin apparatus has a capacity of 10 tons each.

In order to catch the coarse gold which might escape solution in the agitator tank, the pulp from the batteries is first delivered to a series of step amalgamating plates, four plates being provided for each agitator. This auxiliary apparatus is located on the top of the agitator tanks. These amalgamating plates are also kept under the influence of an electric current, which, however, performs no electrolytic function, the object being merely to keep a small current passing through the plates. It is claimed to have been demonstrated by practical experience that this method of procedure keeps the plates soft and prevents them from crystallizing.

The main agitation takes place in a very weak cyanide solution, namely, ¾ pound of cyanide per ton, and it is claimed that on this account it is possible to dissolve the precious metal values without experiencing any detrimental effects from the presence, for instance, of small amounts of copper. It will be very interesting to ascertain the results obtained by this installation when working on such a large scale, although it is claimed that tanks of the same size as those adopted in this plant have been successfully worked on a practical scale, and that the problems that are likely to arise have already been anticipated and solved.

**Mineral Production of United States by States.**—A new feature of the 1905 volume "Mineral Resources of the United States" of the United States Geological Survey is a series of tables, showing the value by States of the mineral products of the country. These include both certain raw materials and also products, for example, both iron ores and pig iron, both pig lead and lead paints, etc., regardless of the consequent duplication of values. Some of the results of these tables are quite surprising. For example, we think of Colorado and California as our most representative mineral States, and yet the actual value of Illinois mineral products (\$105,065,567) was far greater than that of either Colorado (\$59,280,944) or California (\$43,406,258) last year. The list is headed by Pennsylvania (\$569,828,673) and Ohio (\$169,203,710). Illinois is third.

## Water Gas.

(Translated from Baron Hanns Jüptner von Jonstorff's Chemical Technology of Energies.)

BY OSKAR NAGEL, PH. D.

Instead of producing fuel gases by the action of the oxygen in the air on glowing coal, we can use for this purpose the oxygen of water in place of the oxygen in the air.

If steam is led over glowing coal, according to the temperature, two different reactions will take place. At very high temperatures the reaction takes place according to the equation:



while with decreasing temperature a second reaction becomes more and more prevalent according to equation:



The first equation is furnishing a mixture of equal volumes of CO and H<sub>2</sub>: CO 50 per cent by volume and H<sub>2</sub> 50 per cent by volume, while the second reaction, if taking place exclusively, furnishes a gas containing two volumes H<sub>2</sub> for every one volume of CO<sub>2</sub>, hence CO<sub>2</sub> 33.33 per cent by volume and H<sub>2</sub> 66.67 per cent by volume. The thermal value of the first gas per 22.42 liters is 68 Cal., of the second gas 45.4 Cal.

A comparison of the generator (air) gas process with the two water gas processes shows:

	H <sub>2</sub>	CO
(1) $C + \frac{1}{2}(O_2) + 2N_2 = CO + 2N_2$ .....	...	33½
(2) $C + 2H_2O = CO_2 + 2H_2$ .....	66½	...
(3) $C + H_2O = CO + H_2$ .....	50	50

The figures of thermal value refer to the same gas volume in each case, and are well adapted for comparing the qualities of the gases. In case, however, we want to consider the utilization of fuel, we have to refer the thermal values to equal quantities of carbon (equal volumes of CO and CO<sub>2</sub>), and we obtain

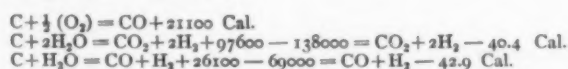
12 Grams C. Yield Liters of Gas.	Value of the Gas at Constant. Volume.	Pressure.
(1) 67.26	67.8 Cal.	68.7 Cal.
(2) 67.26	136.2 Cal.	139.5 Cal.
(3) 44.84	136.0 Cal.	137.0 Cal.

We see that water gas even under the most unfavorable circumstances yields more heat (thermal value) than the ideal air (generator) gas, besides the fact that it contains less non-combustible gases.

For making a perfect comparison we have to calculate at least—if not the pyrometric heating effect—the quantity of air theoretically required for combustion. We have for each 22.42 liters of gas:

Composition of Gas in % by Volume.				Theoretical Amount of Air.		Combustible Indifferent Gases.		Products of Combustion.		
H <sub>2</sub>	CO	CO <sub>2</sub>	N <sub>2</sub>	O <sub>2</sub>	N <sub>2</sub>			H <sub>2</sub> O	CO <sub>2</sub>	N <sub>2</sub>
(1) ...	33½	...	66½	16½	64½	33½	131½	...	33½	66½
(2) 66½	...	33½	...	33½	133½	66½	166½	66½	33½	133½
(3) 50	50	...	...	50	200	100	200	50	50	200

\* As the decomposition of water requires more heat than is furnished by the formation of CO, and even CO<sub>2</sub>, both water gas processes are taking place only with the assistance of external heat. We have:



Considering the external heat we have:

	Thermal Value of Gas per 12 Grams C.	External Heat to be Supplied.	Gain in Heat.
$C + \frac{1}{2}(O_2) + 2N_2 = CO + 2N_2$ .....	68.7 Cal.	-21.1 Cal.	89.8
$C + 2H_2O = CO_2 + 2H_2$ .....	139.5 Cal.	+40.4 Cal.	99.1
$C + H_2O = CO + H_2$ .....	137.0 Cal.	+42.9 Cal.	94.4

The absolute gain in heat is therefore not very great in water gas, and becomes less if we take into consideration losses of heat, which we cannot avoid, when furnishing the external heat to the producer. The advantage of water gas, therefore, does not consist in a gain in heat, but exclusively in the higher thermal value of this gas, which allows a better utilization in the combustion.

As can be seen from the above statements, the reaction  $C + H_2O = CO + H_2$  will take place if steam is led through a layer of sufficiently hot coal. As heat is absorbed by this reaction, the coal will cool off, and besides the above reaction, the process  $C + 2H_2O = CO_2 + 2H_2$  will take place. As the cooling continues the second process will begin to outweigh the first, and finally, since the second reaction also absorbs heat, the coal will be so cold that the reaction will stop, and thus the steam will go through the fuel undecomposed.

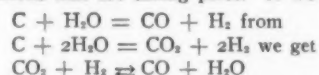
This necessitates reheating the coal in the generator. This is done by shutting off the steam and blowing air through the generator until the coal is sufficiently hot. During this period air (generator) gas is produced which can be utilized independent of the water gas. This period is called "hot-blowing." As soon as the coal is hot again, the air blast is stopped and the steam valve opened, and water gas is made until the cooling off of the fire prevents the rational production of water gas.

Vol. %	Thermal Value of Volume Cal.	Of Mixture at Constant Pressure Cal.
N <sub>2</sub> 66½	22.6	22.9
CO 33½	45.4	46.5
H <sub>2</sub> 50	68.0	68.5

We, therefore, have here an intermittent process, which not only requires careful supervision but also the erection of double the number of generators in places where a continuous stream of water gas is required, and where a large gas holder is objectionable.

As we have seen, the two water gas reactions are taking place in parallel. Since, however, the one furnishes a superior gas with better utilization of coal than the other, it is of importance to know the conditions which determine to which extent each of the two reactions will take place. For this purpose we have to study the state of equilibrium between the two reactions.

To find the equilibrium of the gas phase, we have to consider the reactions that are taking place. If we deduct



This is a reversible reaction in which two volumes (CO + H<sub>2</sub>O) are formed from two volumes (CO<sub>2</sub> + H<sub>2</sub>). It is, there-

fore, independent of pressure at all temperatures above the boiling point of water. One might now conclude that the composition of water gas at a given temperature is independent of the pressure; this, however, is not correct. From the last equation we get for the isothermic equilibrium:

$$K = \frac{CO_2 \cdot CH_2O}{CO_2 \cdot CH_2} \text{ or } \frac{CO}{CO_2} = K_1 \frac{CH_2}{CH_2O}$$

We, therefore, see that at a given temperature there is corres-

ponding to every  $\frac{CO}{CO_2}$  a different  $\frac{H_2}{H_2O}$ . To reach definite

results we have to look for a reaction which determines the equilibrium between the gas phase (in our case consisting of



$\text{CO}_2$ ,  $\text{CO}$ ,  $\text{H}_2$  and  $\text{H}_2\text{O}$ ) and the solid phase (C), and as such we are going to use the equation mentioned already in the generator gas process:



$$K_2 = \frac{(\text{CO})^2}{\text{CO}_2}$$

And now the conditions are given for calculating the isothermic equilibrium. As the last mentioned reaction depends on the pressure, we must necessarily conclude that the composition of water gas also depends on the pressure.

We are going to discuss now the theory of the water gas process in a few words. If we express the steam pressure by  $P$  and the gasifying temperature (in  $^{\circ}\text{C}.$ ) with  $t$ , the ideal composition of the water gas (i. e., the composition corresponding to the equilibrium reached) is as follows:

Vol. %	Steam Pressure, $P$ , in Atmospheres.											
	0.1	0.25	0.5	0.75	1.0	1.5	2.0	2.5	3.0	4.0	5.0	10.0
$t = 400^{\circ}\text{C}.$												
CO.....	0.24	0.12	0.06	0.04	0.03	0.02	0.02	0.01	0.01	0.01	0.01	0.00
CO <sub>2</sub> .....	10.88	7.86	5.97	5.04	4.46	3.73	3.27	2.97	2.73	2.40	2.15	1.55
H <sub>2</sub> .....	21.99	15.84	11.99	10.12	8.94	7.48	6.56	5.94	5.47	4.82	4.31	3.11
H <sub>2</sub> O.....	66.88	76.18	81.98	84.80	86.57	88.77	90.15	91.08	91.79	92.77	93.53	95.34
$t = 600^{\circ}\text{C}.$												
CO.....	26.66	18.87	14.65	11.56	10.03	8.14	6.99	6.20	5.61	4.78	4.22	2.87
CO <sub>2</sub> .....	12.84	16.06	17.15	17.79	17.86	17.67	17.41	17.12	16.84	16.32	15.88	14.32
H <sub>2</sub> .....	52.34	50.89	48.95	47.14	45.75	43.48	41.81	40.44	39.29	37.42	35.99	31.51
H <sub>2</sub> O.....	8.16	14.18	19.25	23.51	26.36	30.71	33.79	36.24	38.26	41.48	43.91	51.30
$t = 800^{\circ}\text{C}.$												
CO.....	49.04	47.81	46.04	44.46	43.06	40.56	38.53	36.83	35.41	32.81	30.60	24.38
CO <sub>2</sub> .....	0.50	1.13	2.02	2.80	3.48	4.66	5.59	6.34	6.95	8.02	8.88	11.05
H <sub>2</sub> .....	50.03	50.07	50.08	50.06	50.02	49.88	49.71	49.51	49.21	48.85	48.45	46.48
H <sub>2</sub> O.....	0.43	0.99	1.86	2.68	3.44	4.90	6.17	7.32	8.43	10.32	11.98	18.09
$t = 1000^{\circ}\text{C}.$												
CO.....	50.00	50.00	50.00	50.00	50.00	49.42	49.42	49.00	48.57	48.35	47.98	46.24
CO <sub>2</sub> .....	.....	.....	.....	.....	.....	0.25	0.25	0.45	0.61	0.71	0.87	1.59
H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	49.92	49.92	49.90	49.79	49.77	49.72	49.42
H <sub>2</sub> O.....	.....	.....	.....	.....	.....	0.41	0.41	0.65	1.03	1.17	1.43	2.75
$t = 1200^{\circ}\text{C}.$												
CO.....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	49.32	49.31	49.31
CO <sub>2</sub> .....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.25	0.25	0.25
H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	49.82	49.80	49.80
H <sub>2</sub> O.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	0.61	0.64	0.64
$t = 1400^{\circ}\text{C}.$												
CO.....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00
CO <sub>2</sub> .....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00
H <sub>2</sub> O.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....

Figs. 1 and 2 show the ideal composition of water gas at a steam pressure of one and four atmospheres. We see from the diagrams that with increasing pressure the curves are moving towards higher temperatures. We also see that the quantity of undecomposed steam present is rapidly decreasing from a certain temperature on, while the quantity of  $\text{CO}$  and  $\text{H}_2$  is rapidly increasing in the same manner. The curves of  $\text{CO}$  and  $\text{H}_2$  are in their middle part practically parallel, but the upward movement of the  $\text{H}$ -curve is beginning  $200^{\circ}\text{C}.$  below the bend of the  $\text{CO}$  curve.

The  $\text{CO}_2$  curve starts to rise together with the  $\text{H}$  curve (but more slowly), until it crosses the steam curve and falls with the latter. The result of this discussion for the practice is, that the most favorable gasifying temperature is between temperature limits of about  $200^{\circ}$ , and increases with the steam pressure.

This becomes clearer when we calculate the quantity of combustible gases ( $\text{CO}$  and  $\text{H}_2$ ) present in water gas (Fig. 3).

As the combustion of one Mol  $\text{CO}$  yields 68,600 Cal., the combustion of one Mol  $\text{H}_2$  to liquid water 68,400 Cal., which is practically the same amount of heat, we can use the above table for comparing the thermal value of the different gases.

Quantity of combustible gases present in ideal water gas:

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	22.23	79.00	99.07	100.00	100.00	100.00
0.25.....	15.96	69.76	97.88	100.00	100.00	100.00
0.5.....	12.05	63.60	96.12	100.00	100.00	100.00
0.75.....	10.16	58.70	94.52	100.00	100.00	100.00
1.0.....	8.97	55.78	93.08	100.00	100.00	100.00
1.5.....	7.50	51.62	90.44	99.34	100.00	100.00
2.0.....	6.58	48.80	88.24	99.34	100.00	100.00
2.5.....	5.95	46.64	86.34	98.90	100.00	100.00
3.0.....	5.48	44.90	84.62	98.36	100.00	100.00
4.0.....	4.83	42.20	81.66	98.12	99.14	100.00
5.0.....	4.32	40.21	79.14	97.70	99.11	100.00
10.0.....	3.11	34.38	70.86	95.66	99.11	100.00

As one Mol of every gas at  $0^{\circ}$  and 760 min. pressure occupies a space of 22.42 liters, we can calculate the thermal value of

1 cubic meter of the above gases in large Calories by multiplying their content of combustible gases with

$$\frac{1000 \times 68.5}{100 \times 22.42} = 30.6$$

Thermal value of 1 cubic meter of ideal water gas in large Calories:

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	680	2417	3032	3060	3060	3060
0.25.....	590	2135	2995	3060	3060	3060
0.5.....	369	1946	2941	3060	3060	3060
0.75.....	311	1715	2892	3060	3060	3060
1.0.....	274	1707	2848	3060	3060	3060
1.5.....	230	1580	2767	3040	3060	3060
2.0.....	201	1493	2700	3040	3060	3060
2.5.....	182	1427	2642	3026	3060	3060
3.0.....	168	1374	2589	3010	3060	3060
4.0.....	148	1353	2499	3002	3034	3060
5.0.....	132	1230	2422	2990	3033	3060
10.0.....	95	1052	2168	2927	3030	3060

This table shows more clearly that the thermal value of the ideal water gas increases with increasing temperature and decreases with increasing pressure.

At a steam pressure of 1 to 2 atmospheres the most favorable gasifying temperature is between 800° to 1,000° C., and at 10 atmospheres pressure between 1,000° and 1,300° C. It is, therefore, not advisable to use steam of too high pressure.

The quality of the water gas is deteriorated by its content of undecomposed steam and of CO<sub>2</sub>. We, therefore, will con-

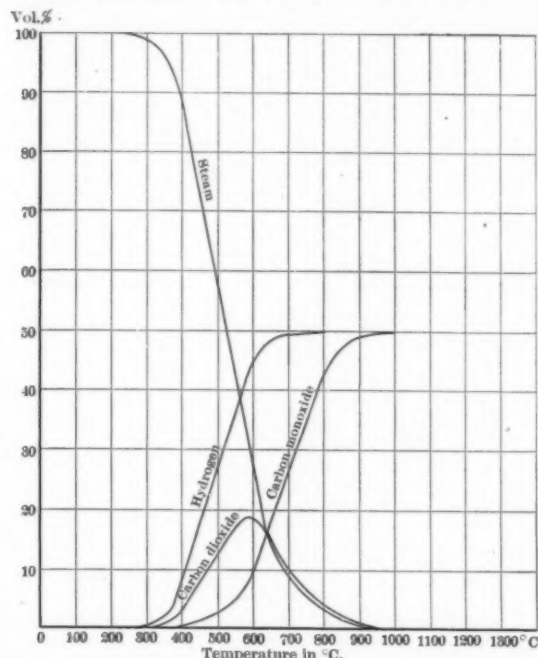


FIG. 1.—IDEAL COMPOSITION OF WATER GAS IN PER CENTS BY VOLUME AT 1 ATMOSPHERE PRESSURE.

sider the influence of pressure and temperature on the quantity of H<sub>2</sub>O and CO<sub>2</sub> present in the gas.

The quantity of undecomposed steam in the ideal water gas decreases rapidly (Fig. 4) with increasing gasifying temperature and slowly increases with the pressure. As thereby the inflammability of the gas is decreased, the gasifying temperature should not be below 700° to 800° C. with a steam pressure of 1 to 10 atmospheres, since otherwise the quantity of undecomposed steam will be considerably above 10 per cent by volume.

The content of CO<sub>2</sub> (Fig. 5) is injurious, as it causes an unfavorable utilization of the carbon. Moreover, it deteriorates the gas, increasing the quantity of non-combustibles and lowering the temperature of combustion. As the CO<sub>2</sub> amounts only to a few per cents at 600° to 700° C., it has not to be considered in the production of generator gas.

In practice, however, it is of importance to know the quantities of carbon and steam which are required for the formation of 1 cubic meter of water gas. This information is given in the following tables:

Quantity of steam in m<sup>3</sup> required for the formation of 1 m<sup>3</sup> of ideal water gas:

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	0.8887	0.6050	0.5046	0.5000	0.5000	0.5000
0.25.....	0.9202	0.6057	0.5106	0.5000	0.5000	0.5000
0.5.....	0.9397	0.6820	0.5194	0.5000	0.5000	0.5000
0.75.....	0.9492	0.7065	0.5274	0.5000	0.5000	0.5000
1.0.....	0.9551	0.7211	0.5346	0.5000	0.5000	0.5000
1.5.....	0.9625	0.7419	0.5478	0.5033	0.5000	0.5000
2.0.....	0.9671	0.7560	0.5588	0.5033	0.5000	0.5000
2.5.....	0.9702	0.7668	0.5683	0.5505	0.5000	0.5000
3.0.....	0.9726	0.7755	0.5764	0.5092	0.5000	0.5000
4.0.....	0.9759	0.7890	0.5917	0.5094	0.5043	0.5000
5.0.....	0.9784	0.7990	0.6043	0.5115	0.5044	0.5000
10.0.....	0.9845	0.8280	0.6457	0.5217	0.5044	0.5000

Therefore 1 cubic meter of steam furnishes the following numbers of cubic meters of ideal gas:

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	1.125	1.653	1.981	2.000	2.000	2.000
0.25.....	1.087	1.537	1.958	2.000	2.000	2.000
0.5.....	1.068	1.466	1.925	2.000	2.000	2.000
0.75.....	1.053	1.415	1.896	2.000	2.000	2.000
1.0.....	1.047	1.386	1.871	2.000	2.000	2.000
1.5.....	1.039	1.348	1.825	1.986	2.000	2.000
2.0.....	1.034	1.323	1.789	1.986	2.000	2.000
2.5.....	1.031	1.304	1.759	1.978	2.000	2.000
3.0.....	1.028	1.289	1.735	1.963	2.000	2.000
4.0.....	1.024	1.269	1.690	1.963	1.983	2.000
5.0.....	1.022	1.251	1.655	1.955	1.982	2.000
10.0.....	1.015	1.208	1.548	1.916	1.982	2.000

The last table is specially valuable for this practice, since it permits an easy control of the operation of the generator and allows the determination of the ideal gasifying temperature, which corresponds to the process. The content of one component of the gas, for instance CO<sub>2</sub> (which can be easily determined with an Ados or Strache apparatus) being known, the complete analysis of the gas can be found.

One cubic meter of water gas contains grams of C.:

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	59.51	211.40	265.14	267.60	267.60	267.60
0.25.....	42.71	186.95	261.23	267.60	267.60	267.60
0.5.....	32.27	170.19	257.22	267.60	267.60	267.60
0.75.....	27.19	157.08	252.94	267.60	267.60	267.60
1.0.....	24.03	149.27	249.08	267.60	267.60	267.60
1.5.....	20.07	138.14	242.07	265.83	267.60	267.60
2.0.....	17.61	130.59	236.13	268.83	267.60	267.60
2.5.....	17.05	124.81	231.05	264.66	267.60	267.60
3.0.....	14.66	120.15	226.71	263.21	267.60	267.60
4.0.....	12.90	112.93	218.52	262.57	265.30	267.60
5.0.....	11.56	107.58	211.78	261.45	265.25	267.60
10.0.....	8.30	91.96	189.62	255.99	265.25	267.60

One cubic meter of steam gasifies grams of C. (Fig. 6):

Steam Pressure in Atm.	400	600	800	1000	1200	1400
0.1.....	66.96	349.44	525.44	535.20	535.20	535.20
0.25.....	46.41	287.30	511.61	535.20	535.20	535.20
0.5.....	34.81	249.54	495.23	535.20	535.20	535.20
0.75.....	28.64	222.34	479.59	535.20	535.20	535.20
1.0.....	25.16	207.00	465.92	535.20	535.20	535.20
1.5.....	20.85	186.19	441.89	528.17	535.20	535.20
2.0.....	18.21	172.74	420.77	528.17	535.20	535.20
2.5.....	17.57	162.77	406.54	523.56	535.20	535.00
3.0.....	15.07	154.93	393.32	516.91	535.20	535.00
4.0.....	13.22	143.13	369.31	511.52	526.17	535.20
5.0.....	11.81	134.64	350.45	511.14	525.87	535.20
10.0.....	8.43	115.06	293.67	490.68	525.87	535.20

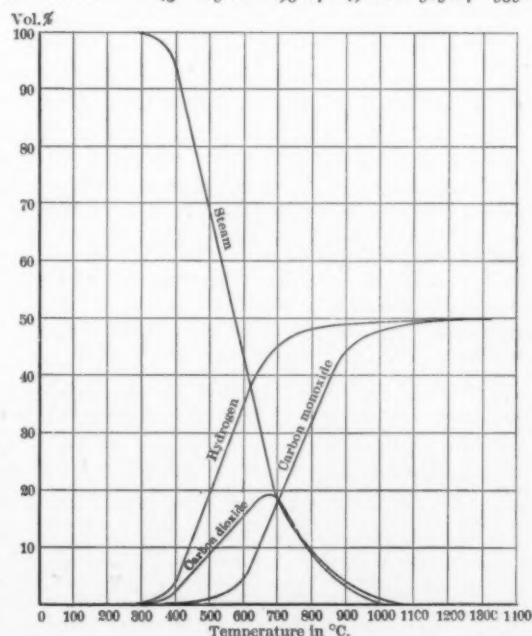


FIG. 2.—IDEAL COMPOSITION OF WATER GAS AT 4 ATMOSPHERES PRESSURE.

If the steam of the gas condenses—which frequently happens in practice—the composition and thermal value of the gas changes accordingly. The calculation of the gas composition from the  $\text{CO}_2$  content is very simple. The  $\text{CO}_2$  of the dry gas being  $c$  per cent by volume, the content of

$$\text{CO} = 50 - \frac{3}{2} c \% \text{ by volume}$$

$$\text{H}_2 = 50 + \frac{1}{2} c \% \text{ by volume}$$

For example, we take a gas made at  $800^\circ \text{C}$ . and 2.5 atmospheres steam pressure. The  $\text{CO}_2$  content having been found as 6.84 per cent by volume, the content of

$$\text{CO} = 50 - 1.5 \times 6.84 = 39.74 \% \text{ by volume}$$

$$\text{H}_2 = 50 + 0.5 \times 6.84 = 53.42 \% \text{ by volume}$$

The following two tables contain the most important data on dry water gas. Compared with the wet gases, in which at constant pressure the  $\text{CO}_2$  content at first increases with the temperature up to a maximum and then decreases, the dry gases have far more regular properties. The  $\text{CO}_2$  content at

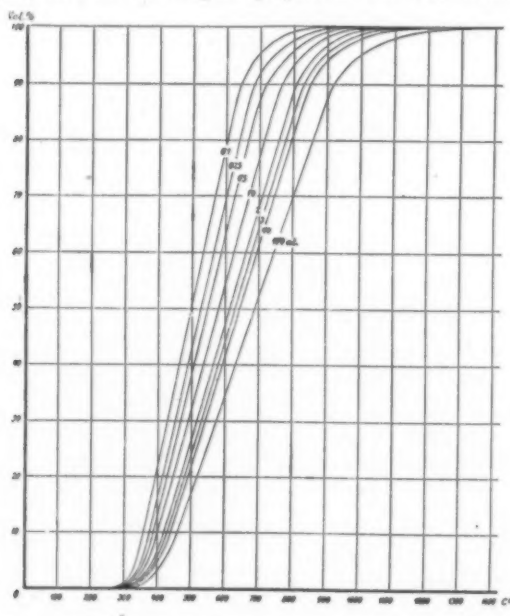


FIG. 3.—CONTENT OF COMBUSTIBLE GASES IN IDEAL WATER GAS.

constant pressure decreases with increasing temperature, while  $\text{CO}$  and  $\text{H}_2$  increase simultaneously. On the other hand  $\text{CO}_2$  increases at constant temperature with the pressure, while  $\text{H}_2$  and  $\text{CO}$  decrease simultaneously.

The following table shows the cubic meters of dry gas produced from 1 cubic meter of steam:

Steam Pressure in Atm.	One M <sup>3</sup> of Steam is Yielding at the Temperatures Stated Below, M <sup>3</sup> Dry Water Gas.					
	400°C.	600°C.	800°C.	1000°C.	1200°C.	1400°C.
0.1.....	0.373	1.518	1.972	2.000	2.000	2.000
0.25.....	0.259	1.304	1.939	2.000	2.000	2.000
0.5.....	0.192	1.184	1.889	2.000	2.000	2.000
0.75.....	0.160	1.082	1.845	2.000	2.000	2.000
1.0.....	0.141	1.021	1.807	2.000	2.000	2.000
1.5.....	0.117	0.934	1.736	1.978	2.000	2.000
2.0.....	0.102	0.876	1.679	1.978	2.000	2.000
2.5.....	0.092	0.831	1.630	1.965	2.000	2.000
3.0.....	0.084	0.796	1.589	1.943	2.000	2.000
4.0.....	0.074	0.743	1.516	1.940	1.971	2.000
5.0.....	0.066	0.702	1.457	1.927	1.969	2.000
10.0.....	0.047	0.588	1.268	1.863	1.969	2.000

The most favorable conditions for producing the dry water gas are therefore the same as for the wet gas. We have so far

discussed the case in which the state of equilibrium is actually reached in the producer. We are now going to consider the case which is very common in practice, that the equilibrium is not reached.

If steam is blown through a layer of glowing coal the re-

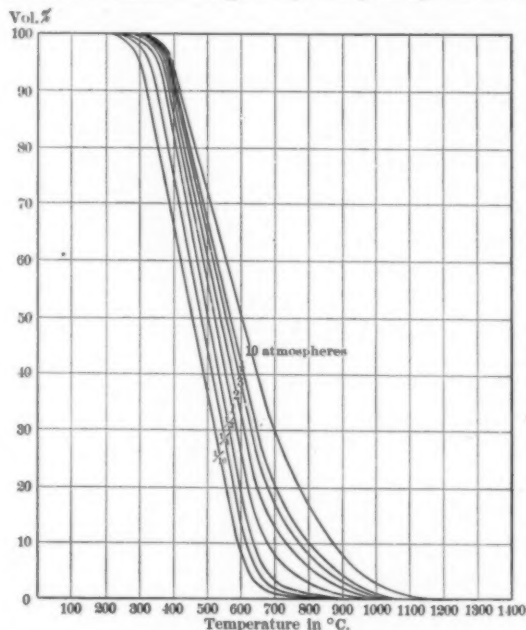


FIG. 4.—CONTENT OF UNDECOMPOSED STEAM IN IDEAL WATER GAS.

action will undoubtedly take place completely on the contact points of steam and coal, i. e., the state of equilibrium will soon be reached here. On his further way the gas current will undergo a change in two respects. Partly by diffusion, partly by mechanical mixture, a reaction will take place between the outer part of the current and the inner part, which is richer in steam; on the other hand, the equilibrium of the outer layer will be disturbed by the contact of same with other parts of the coal.

If the gas passes from the cold to the hot coal layers ("Geg-

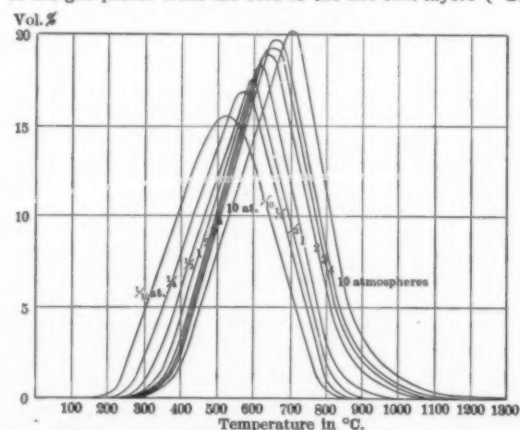


FIG. 5.—CONTENT OF CARBON DIOXIDE IN IDEAL WATER GAS.

enström"), a gas rich in  $\text{CO}_2$  will be formed at first in the outer layer; then, by coming in contact with hot coal, it is enabled to oxidize new quantities of coal, getting thereby richer in  $\text{CO}$ . If, however, the steam passes from the hot to the cold coal layers ("Parallelström"), a gas rich in  $\text{CO}$  will be formed at first in the outer layer, and by passing further it will get richer in  $\text{CO}_2$  and poorer in  $\text{CO}$ .



1 Cubic Meter Dry Gas.	Pressure of the Rejected Steam in Atmospheres.										
	0.1	0.25	0.5	0.75	1.0	1.5	2.0	2.5	3.0	4.0	5.0 10.0
<i>t</i> = 400° C.											
Vol. % CO.....	0.72	0.50	0.30	0.26	0.20	0.18	0.18	0.15	0.14	0.12	0.11 .....
" CO <sub>2</sub> .....	32.85	33.00	33.13	33.16	33.20	33.21	33.21	33.23	33.24	33.25	33.26 33.33
" H <sub>2</sub> .....	66.43	66.50	66.57	66.58	66.60	66.61	66.61	66.62	66.62	66.63	66.63 66.67
" Combustible gas.....	67.15	67.00	66.84	66.84	66.80	66.79	66.79	66.77	66.76	66.75	66.74 66.67
Thermal value.....	2054	2050	2046	2045	2044	2044	2044	2043	2043	2042	2042 2040
<i>t</i> = 600° C.											
Vol. % CO.....	29.00	21.95	18.14	15.11	13.62	11.74	10.55	9.72	9.09	8.00	7.52 5.90
" CO <sub>2</sub> .....	14.00	18.70	21.24	23.26	24.25	25.51	26.30	26.85	27.27	27.84	28.31 29.40
" H <sub>2</sub> .....	57.00	59.31	60.62	61.31	62.13	62.75	63.15	63.43	63.64	64.00	64.16 64.70
" Combustible gas.....	86.00	81.26	78.76	76.42	75.75	74.49	73.70	73.15	72.73	72.00	71.68 70.60
Thermal value.....	2632	2487	2410	2338	2318	2279	2255	2238	2226	2203	2193 2160
<i>t</i> = 800° C.											
Vol. % CO.....	49.20	48.29	46.91	45.66	44.60	42.65	41.06	39.74	38.61	36.09	34.88 29.76
" CO <sub>2</sub> .....	0.53	1.14	2.06	2.89	3.60	4.90	5.96	6.84	7.59	8.94	10.08 13.49
" H <sub>2</sub> .....	50.27	50.57	51.03	51.45	51.80	52.45	52.98	53.42	53.80	54.97	55.04 50.75
" Combustible gas.....	99.47	98.86	97.94	97.11	96.40	95.10	94.04	93.16	92.41	91.06	89.92 86.51
Thermal value.....	3049	3025	2997	2972	2950	2910	2906	2851	2828	2786	2752 2645
<i>t</i> = 1000° C.											
Vol. % CO.....	50.00	50.00	50.00	50.00	50.00	49.62	49.62	49.32	49.07	48.92	48.68 47.55
" CO <sub>2</sub> .....	0.00	0.00	0.00	0.00	0.00	0.25	0.25	0.45	0.62	0.72	0.88 1.63
" H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	50.13	50.13	50.23	50.31	50.36	50.44 50.82
" Combustible gas.....	100.00	100.00	100.00	100.00	100.00	99.75	99.75	99.55	99.38	99.28	99.12 98.37
Thermal value.....	3060	3060	3060	3060	3060	3052	3052	3046	3041	3038	3033 3010
<i>t</i> = 1200° C.											
Vol. % CO.....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	49.62	49.62 49.62
" CO <sub>2</sub> .....	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.25	0.25 0.25
" H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.13	50.13 50.13
" Combustible gas.....	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	99.75	99.75 99.75
Thermal value.....	3060	3060	3060	3060	3060	3060	3060	3060	3060	3052	3052 3052
<i>t</i> = 1400° C.											
Vol. % CO.....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00 50.00
" CO <sub>2</sub> .....	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00 0.00
" H <sub>2</sub> .....	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00	50.00 50.00
" Combustible gas.....	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00	100.00 100.00
Thermal value.....	3060	3060	3060	3060	3060	3060	3060	3060	3060	3060	3060 3060

We will now consider again the reaction between the outer gas layer and the inner steam current. In working according to the "Gegenstrom principle," the steam of the inner surface can react with the outer gas layer, so that CO is oxidized to CO<sub>2</sub> and H<sub>2</sub> is liberated. Supposing the temperature remains constant or decreases, the thermal value of the gas remains unchanged. If, however, the average temperature of the gas current rises—which is probable, since the gas comes into the hotter parts of the producer—this reaction decreases and the actually occurring improvement in the quality of the gas cannot be explained but by oxidation of glowing coal by means of the CO<sub>2</sub> and the steam of the outer layer and also by the outward diffusion of the steam.

If we work according to the "Parallelstrom principle" the hot outer layer formed in the start will react vigorously on the steam (on account of the higher temperature both the diffusion and velocity of reaction will be greater) and the gas without practical change in thermal value will get richer in H<sub>2</sub> and poorer in CO. Hereby the quality of the gas is improved, just the same as above, by the reaction of the outwardly diffusing steam with the glowing coal. On the other hand, the gas quality is deteriorated as the steam gradually comes in contact with cooler coal, whereby the quantity of CO<sub>2</sub> is increased.

Undoubtedly the first mentioned way of gasifying is more advantageous, the more so as herein the gas and steam current is also preheated gradually.

If we consider the average composition of water gas, in case the state of equilibrium is not reached, we always find this relation between CO<sub>2</sub>, CO and H<sub>2</sub>, that the volume of H<sub>2</sub> is equal to the sum of the CO volume and double the CO<sub>2</sub> volume. Besides this some steam is also present. The composition of the wet water gas as well as of the dry gas will, therefore, under all conditions correspond to one equilibrium, which, however, at the same steam pressure corresponds to another

("the ideal") gasifying temperature, the latter being—similar as with generator gas—lower than the actual gasifying temperature.

Dr. Hugo Strache and R. Jahoda have studied the influence of height of fuel and air and steam velocity on this process, both during hot-blowing and gas making, and have found:

In the beginning of the hot-blowing period (when the temperature of the fuel is rather low) CO<sub>2</sub> is formed almost exclusively without any CO, while with increasing temperature the formation of CO increases. We have here again the equilibrium which was mentioned before:  $2\text{CO} \rightleftharpoons \text{CO}_2 + \text{C}$ .

As less C is absorbed by a certain volume of air for the formation of CO<sub>2</sub> than for the formation of CO, the fuel consumption is considerably less in the first stages of hot-blowing

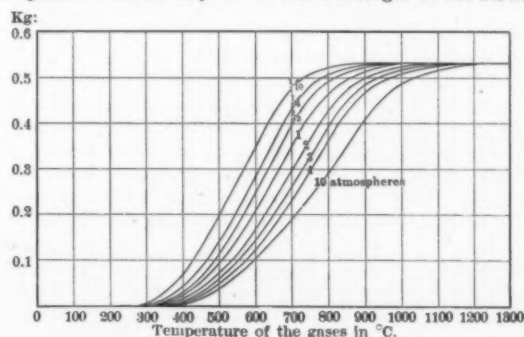


FIG. 6.—KILOGRAMS OF CARBON GASIFIED BY 1 CUBIC METER STEAM.

than in the later stage, while the quantity of heat developed per minute is very much greater in the start than in the later stages.

The loss of heat by the hot gas leaving the producer increases with the temperature. The heat accumulated in the producer

is evidently equal to the difference of generated and lost heat. The ratio of accumulated heat and carbon used is called by Strache "the efficiency in hot-blowing." This ratio is high in the beginning (at low temperature) and decreases with increasing temperature and fuel consumption.

Content of  $\text{CO}_2$  and efficiency in hot-blowing are as follows at

	Efficiency.	$\text{CO}_2$
625° C.....	80%	18 %
672° C.....	70	16
929° C.....	40	7.6
1300° C.....	30	4.6

The total efficiency for a certain blowing period decreases rapidly between 650° and 900°, it is therefore advantageous not to raise the temperature and the producer above 900°.

The losses of heat during the hot-blowing period can be utilized to a large extent for preheating the steam (in the manufacture of pure water gas).

The losses during gas making depend on the velocity of steam and the temperature of the producer. Too low velocity yields a rather small quantity of gas and causes comparatively great loss of heat by radiation; too great velocity is disadvantageous on account of the steam going through undecom-

gas making, and from the loss of heat during blowing and gas making.

Fig. 7 shows a diagram of these conditions.

The total efficiencies also show a maximum at a certain velocity of steam.

At 780° C.....	72.5 per cent.
At 860° C.....	77 per cent.

### The Electric Furnace in Iron and Steel Metallurgy

The interest which the subject has at present is indicated by the number of papers which are being written concerning it. Five of special interest are herewith abstracted.

Dr. R. S. HUTTON, of Owens College, Manchester, England, recently discussed the matter in an able paper before the Sheffield Society of Engineers and Metallurgists. The introduction of his paper contains some interesting data on the cost of power. Several figures are given on cheap water power, varying all the way from \$20 at Niagara Falls down to an alleged rate of \$3 per horse-power-year in Norway.

In discussing steam power the results of electric central stations are generally taken into consideration, but the author points out how wrong this is for electrometallurgical work. The question of power generation on a large scale, for continuous use at full load and on the spot, without the cost of distribution, is entirely different from central station practice. There is no doubt that the subject has been greatly prejudiced by invoking, for the sake of argument, the costs of municipal and other stations, few of which attain more than a 30 per cent load factor. This is particularly important in estimating the capital costs (interest and depreciation) per unit.

As to what can be done in practice, "there seems to be every possibility that in a power plant properly designed for the purpose, and with a small number of large units, in favorable situations, with cheap coal, and with a good water supply, the kw-hour can be obtained by steam at the total inclusive cost of about 0.5 cent." "There are in this country (England) several large private plants running under these conditions, and so far as can be gathered from the evidence of those connected with them (see *e. g.*, Minutes of Evidence of Royal Commission on Coal Supplies), the actual cost in practice is not materially above this figure."

From figures given by the Power Gas Corporation the author states that in the case of a Mond ammonia-recovery gas plant working continuously with a steady load, with coal at \$2 per ton (the gas being used for power and heating purposes), the total cost of electricity (excluding interest and depreciation) produced by means of six 300-hp. gas-dynamos, amounted to only 0.2 cent per kw-hour. The same company, as a result of its extensive experience, estimates the cost with a 3,000-kw. plant, working 132 hours per week, and with coal at \$2 per ton, at 0.126 cent per kw-hour total working costs, or 0.210 cent total cost, inclusive of interest and depreciation charges. "Whilst the latter figures may need to be taken with some reserve, it must be remembered that in this country (England) there are several electrochemical works with a steady and continuous load in which large gas-engine installations have been adopted in place of a previously equipped steam plant working under very economical conditions."

With regard to the problem of iron ore reduction in the electric furnace the author gives a review of the results obtained at Sault Ste. Marie (our Vol. IV., pp. 124, 265, 332). "Although these developments have no immediate bearing on our own (England's) industries, they most certainly deserve the attention of those who are interested in the development of our colonies, and practically as in Canada far too much is left to the enterprise and capital of the United States."

With regard to steel processes the author first gives a review of the development of the Kjellin induction furnace, and then

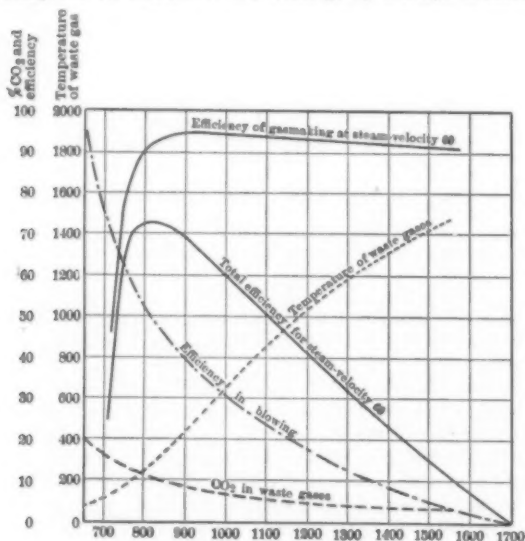


FIG. 7.—EFFICIENCY CURVES.

posed; in this case large quantities of heat leave the producer without being utilized on account of the high specific heat of steam.

The results of these researches are:

- (1) The quantity of undecomposed steam and the  $\text{CO}_2$  content of the gas increase at constant temperature with the increasing velocity of the steam in about the same proportion.
- (2) The content of steam and  $\text{CO}_2$  of the crude gas at constant velocity of steam decrease with increasing temperature.
- (3) Even at low temperature the content of  $\text{CO}_2$  and steam can be reduced to a minimum by decreasing the velocity of steam.

The efficiency in gas making is calculated from the carbon consumption during gas making, loss of heat in the producer and the thermal value of the water gas produced. The loss of heat is composed of the heat of formation, heat of the gas produced and of the undecomposed steam and the radiation of heat from the producer. For every temperature there is a certain velocity of steam with which a maximum efficiency is reached (87 per cent to 93 per cent).

The total efficiency for any given velocity of steam can be calculated from the carbon consumption during blowing and

speaks on the Héroult furnace. Concerning the latter he makes the following remarks:

"The very fact that the process seems so contrary to general practice merely shows that it is attempting something which has previously been impossible. The inventions of Bessemer and Thomas and Gilchrist appeared in exactly the same light to the vast majority of our manufacturers; but, nevertheless, some were found to carry the invention forward to practical success. Should the Héroult steel process succeed in justifying its claims, there is no doubt that it would be the greatest advance in the general principles of steel manufacture which has been brought forward since the introduction of the basic system."

After a brief description of the process the author points out that the two main features of this process which call for special attention are the purification and the recarburization. In considering the possibility of completely removing phosphorus and sulphur by the action of basic slags, care should be taken to note how very different are the conditions of working from those which exist in the open-hearth process. Here a comparatively thin layer of slag is brought to an exceedingly high temperature. This highly-superheated slag and the metal immediately underneath it are brought into intimate contact, and also caused to rapidly circulate by the convection currents which are set up as a result of the large quantity of heat rapidly transmitted to them. In these facts and in the system of renewing the slag at intervals will probably be found the explanation for the very low values to which the phosphorus can be brought.

The recarburization and the final condition of the steel are also very different from what is met with in the open-hearth process. In its present form the Héroult furnace is so completely protected from ingress of air that the basic slag in the finishing process contains quite a considerable amount of calcium carbide. This not only proves the high temperature to which the slag is taken, but also gives very strong evidence for the absence of air. The whole atmosphere in the furnace above the slag consists of practically pure carbon monoxide and the regular production of metal free from dissolved oxide, and with the minimum of occluded gas, is not only possible but is actually accomplished in practice.

The paper is concluded by a few notes on the steel furnaces of Keller, Girod and Stassano.

Two papers on the electric furnace in steel metallurgy were presented before the Verein Deutscher Eisenhüttenleute at their recent December meeting. They are abstracted here from the *Iron and Coal Trades Review*.

Dr. EICHNOFF spoke on the Héroult process as operated at Remscheid, Germany (our Vol. IV., p. 363). He stated that in the largest furnace yet built there for 5-ton charges, the power required to produce a ton of steel still amounted to 750 kw-hours with a cold charge of scrap. If, however, the raw material was melted according to the open-hearth or basic process, and then transferred in a liquid condition to the electrical furnace, the consumption of energy in such a small furnace would be from 200 to 300 kw-hours, according to the desired purity of the material. The consumption in larger furnaces would diminish.

The question was, however, quite different when special steels were concerned. In the case of a tipping open-hearth furnace on the Wellmann system, a charge of from 1½ to 2 tons of liquid steel, which had been more or less purified, was removed, excluding the slag, and placed in the Héroult furnace and then treated. It was possible to make a charge in an hour and a quarter corresponding to a power consumption of 200 kw-hours per ton of steel.

The great heat of the arc, in the opinion of the author, was the ground for its being possible to effect the far-reaching purification and deoxidation, and the original apprehension that the heat might injure the steel had not been fulfilled. The

bath was always in brisk circulation, and the individual parts of the same were only exposed to the high temperature for a short time.

As to the works at Remscheid, the plant was started on Feb. 17, 1906, and the demand for the steel produced had become so large that it had been necessary to arrange for the doubling of the installation. As a consequence, an order had been given for a new generating plant of 1,000 hp., a second furnace and a large rolling mill.

Hitherto interruptions had occurred in the Wellmann furnace used for melting purposes, although these could be obviated by the necessary experience. The electric furnace was at work to-day with the same hearth as was built in at the beginning.

As far as working expenses were concerned, the author mentioned that the furnace required two men and a youth, but if cold material was employed it was necessary to add the services of from one to two chargers, according to the size of the furnace. The electrode consumption cost from \$0.75 to \$1.00, according to the furnace size with a cold charge, and from 25 to 60 cents with a liquid charge, while the consumption of limestone and ore was not greater than in other processes. The use of ferro-manganese and ferro-silicon was largely economized, and the cost of repairs and fireproof material was much less than in the case of an open-hearth furnace.

Mr. ROECHLING then spoke on the induction furnace, and stated that it is possible to remove in the same at will and completely sulphur, phosphorus, etc., in an easy and favorable manner. The peculiar sphere of the process is not melting but rather the improvement of liquid material. "As to the cost of the system, it has repeatedly been possible at Volklingen to completely degasify ordinary basic steel, and to remove phosphorus and sulphur to mere traces, all with a consumption of from 150 to 200 kw-hours per ton. This result was obtained with a small furnace of 600-pound capacity."

\* \* \* \*

Mr. JOH. HÄRDEN presented a paper before the Sheffield Society of Engineers and Metallurgists on the electric induction furnace. After a short review of the history of the subject, in which he mentioned the names Ferranti, Colby and Kjellin, he characterized the induction furnace as a transformer with a single secondary winding. He emphasized very strongly that any arrangement in which the primary coil is so placed that the individual windings do not come as close as possible to both the core and the secondary, must involve a bad power factor, and he claimed that Kjellin, who has patented placing the primary coil right in the center, near to the secondary and close to the core, thus obtains the best power factor for this type of furnace.

As to what can be done in the induction furnace the author stated that the furnace is only a modified crucible furnace. High-grade raw material, as used hitherto, must still be used for producing high-class steel. It is not commercially possible to make high-class steel from inferior material, notwithstanding laboratory assertions to the contrary.

The suitability of the electric induction furnace for producing big crucible quality ingots for crank shafts and axles was then pointed out, and the considerable saving in cost over the ordinary crucible furnace was shown.

\* \* \* \*

Finally, a paper by Mr. J. W. EVANS, presented to the Canadian Mining Institute, shall be recorded, describing some laboratory experiments in which the author endeavored to make steel directly from iron ore.

Several laboratory furnaces are described, and the scale of experiments is indicated by the statement that in the present electric furnaces of the author an ounce of iron ore is reduced in from 15 to 20 minutes and converted into steel.

As to furnace linings, it is said that carborundum or corundum mixed with 8 per cent of tar, when thoroughly baked,



makes a very satisfactory lining, although it must not come in contact with an iron charge. An inside lining of magnesite overcomes this difficulty. Charcoal was used throughout as reducing agent in the experiments and limestone as flux.

"In treating the sulphurous iron ores over 92 per cent of the sulphur was slagged off, and by varying the amount of lime better results should be obtained. In treating the titanium iron ores steel was produced which did not contain a trace of titanium, but as the writer wished to retain a portion of the titanium the quantity of lime in the charge was reduced, and the result showed that the titanium contents in the steel can be readily governed in this way, although the temperature of the furnace will have to be kept as constant as possible."

The electric furnace process is also said to solve the question of treatment of magnetic iron sands, of which there are large deposits in both Ontario and Quebec, and "the writer has made a mild steel containing .05 per cent carbon from magnetic sands from the St. Lawrence River. These sands could be dredged and dried, magnetically separated, and conveyed direct to the furnace at a considerable saving in cost over mining and crushing iron ore."

When much lime is used as flux, titanium slags off readily. If titanium is retained in the steel, by lowering the amount of lime, a large portion of silicon is retained also.

### Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

THE FARADAY SOCIETY.

The first meeting of the winter session of this Society had a distinctly chlorous atmosphere. Not in its physical sense, but in a literary sense. The first two papers both dealt with the electrolytic hypochlorites, and the discussion thereon entirely crowded out any discussion on the third paper. The first paper, by Mr. W. Pollard Digby, upon "Some Investigations Relative to the Depreciation of Electrolytically Produced Solutions of Sodium Hypochlorite" was described in outline by the author, whereupon Dr. F. Mollwo Perkin, who was in the chair, called upon Mr. Charles V. Biggs to read his paper upon the "Hermite Electrolytic Processes at Poplar." (Both papers are abstracted on p. 7 of this issue.—Ed.)

A combined discussion being called for, Mr. Spiers read a communication from Mr. J. B. C. Kershaw of a rather scathing nature. Mr. Biggs had paraphrased in an appendix a few lines from Mr. Kershaw's article in the April issue of *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*. This paraphrase did not suit Mr. Kershaw, who thereupon repeated in exact words the opening column of the article in question, including his theory of the chemical reactions taking place during electrolysis, a theory differing *in toto* from one by Dr. W. R. Hodgkinson (of Woolwich Arsenal), which the author had quoted. Mr. Kershaw closed with an attack upon Poplar administrative extravagances, citing the Hermite installation as a typical sample on account of its poor electrochemical efficiency as compared with that of certain Continental processes.

Dr. R. H. Hutton, who followed, asked for particulars as to the size and shape of electrodes, length and weight of platinum wire, the durability of the zinc, and for one or two tables in Mr. Biggs' reply, giving the relative costs of salt and electricity per kilogramme of available chlorine at various current densities. He remarked that Mr. Kershaw's point as to electrochemical efficiency as the one desideratum was inapposite. Anybody could show high efficiencies with saturated saline solutions, and Mr. Kershaw's figures were for a process using a saturated solution.

Dr. Alexander next made a very pleasing speech, detailing many minor points which had needed attention in working the process. He referred to an earlier controversy between Mr. Kershaw and himself, and scored notably upon the point of cost. In addition to street watering during hot weather, they

had distributed the hypochlorite liquid freely in all cases of infectious diseases. Excluding the cost of labor, which was now actually less than when coal tar disinfectants were purchased in bulk and diluted for distribution, the local authority had only expended £22 in eight months as against £314 for the twelve months of the preceding year. Only unskilled labor was employed. As to the durability of the zinc cathodes, the Netley plant was still at work with its original zinc cathodes. They secured stability at Poplar by adding an excess of caustic soda to the final product, and their liquid was certainly stable. With regard to Mr. Digby's paper, his own view, and also that of Monsieur Hermite, was to the effect that the presence of salt occasioned instability. Monsieur Hermite was now producing, by a secret process, a stable liquid containing a pure hypochlorite of magnesium, and without any chloride whatever.

Dr. Rideal, speaking as a chemist and bacteriologist, gave some reminiscences of the early Hermite and Woolf and recent Atkins' processes in this country, alluding specially to the loss occurring in the presence of easily oxidizable organic matter. He considered that the electrolytic hypochlorite produced *in situ* had a great many advantages in bactericidal activity over chloride of lime.

Further, to quote the cost of chloride of lime per ton free on rails, and to draw deductions from this, was not fair, as it overlooked transport charges, cost of mixing, dealing with sludge, handling of returned empties, etc. Given a suitable electrolyser, what could be simpler for municipal purposes than to turn on a tap, put some salt in a tank, and switch on the current?

Such plants were immediately available for many emergencies. Much could be done by them in regard to street watering. For instance, if the highly deliquescent calcium chloride was used and part of it sent through an electrolyser, it not only laid the dust, but also sterilized it—a very important matter hygienically.

As to the earlier processes, these had certainly been costly, but then the fault was to be found in the large amount of salt which remained undecomposed in the electrolyte. The best process in the future from the chemist's point of view, and also the best commercially, would be that using the smallest quantity of chloride per kilogramme of available chlorine. In the case of the sterilization of a sewage effluent entering a water supply, a minimum of chemical substances should be used. As for Mr. Digby's paper on stability, he was afraid that, despite long investigations, Mr. Digby did not know much about what really caused instability. Nor, for that matter, did anybody else. Perhaps the amount of alkali present was a vital factor in the case. He (Dr. Rideal) was inclined to regard high temperature during manufacture as of little importance.

Dr. Borns, referring to Mr. Digby's paper, asked for information as to how the galvanic couples which had been cited as occasioning exceptional depreciation had been arranged. With regard to the depreciation, due to ebouite, perhaps a sulphide had been formed, which, when present in minute quantities accelerated the depreciation, due to the metallic surface.

Mr. L. A. Smart dealt with the potential value of hypochlorite processes as station-load equalizers, but asked for a less efficient process as consuming more current.

Mr. Wolf Defries emphasized the importance of economy in salt consumption. Salt had been constant in price for quite a long time, and there was no great prospect of a decline. The cost of electrical energy had not yet reached bed-rock prices. As for calcium chloride, which could be had at a fairly cheap rate, they must remember that new uses were being continuously found for so-called "waste products." As for Dr. Alexander's pioneer work at Poplar, he had really rendered a public service, the outcome of which was far more hopeful than the prices mentioned by Mr. Kershaw or Mr. Hutton would indicate.

The authors then being called upon to reply, asked for permission to do so in writing, owing to the lateness of the hour.

Dr. A. C. Cumming's paper on "The Electrochemistry of Lead" was taken and read, discussion being deferred until the next meeting. Dr. Cumming thus secures priority of publication over a Continental scientist who is at work upon the same subject.

#### CASTNER-KELLNER ALKALI CO., LTD.

The report for the half year which ended Sept. 30 last, states the net profit for the last six months is £35,946, to which has to be added the amount brought forward from last account, £14,548, making a total of £50,494, and after deducting debenture and other interest (£4,971) there remains an available balance of £45,523. The directors now recommend that £15,000 be apportioned to depreciation reserve, increasing that account to £130,000, that a dividend at the rate of 7 per cent per annum be paid for the six months, leaving to be carried forward £14,773. The new works at Wallsend-on-Tyne are now working satisfactorily. This marks a very pleasing advance, and indicates that one electrochemical industry at least—the manufacture of alkali and bleaching powders—is flourishing in the districts once regarded as the sole provinces of purely chemical processes.

#### METALLURGICAL PAPERS READ IN NOVEMBER.

The papers read at the meeting of the Institution of Mining and Metallurgy, held on Nov. 15, were two in number. The first, by Mr. T. C. Cloud, was descriptive of the Wallaroo Smelting Works, in South Australia. The installation in addition to the smelting of copper has various subsidiary departments. Copper sulphate works, capable of turning out about 500-600 tons of sulphate per annum; sulphuric acid works, with an output of about 5,000 tons of chamber acid per annum; a department for the smelting of refractory gold ores, using copper as a collecting material, and, combined with this, an electrolytic depositing plant turning out about 12 tons of copper per week; and also an electrolytic plant for parting gold and silver.

The ores treated in the smelting works consisted of, for the most part, chalcopyrite (copper pyrites) mixed with varying proportions of pyrite (iron pyrites). A small proportion of the ore from the Moonta Mines consisted of bornite, and occasional parcels consisting chiefly of copper glance were received. Generally speaking, the ore from the Wallaroo Mine carried a larger proportion of pyrite than that from the Moonta Mine. The chief minerals forming the gangue of the ore are quartz, felspar, biotite, hornblende and calcite, while apatite and scheelite also occur in small quantities. The ore, as received at the works, is either (a) hand-picked and broken to about 2½-inch gauge, or (b) jigged concentrates or slimes and fines, the proportions being about 13 of (a) to 20 of (b).

The ores under the heading (b) are oxidized copper ores, consisting of green carbonate and cuprite, with occasional parcels containing a large proportion of the copper in the form of atacamite and ores consisting chiefly of bornite and copper pyrites, comparatively free from iron pyrites. The oxidized ores here referred to come to hand for the most part as hand-picked ore, broken to about 2½-inch gauge, while the sulphide ores mentioned reach the works as jigged concentrates and slimes.

The paper contains descriptions of the *calcining plant* (which is divided into two sections, one dealing with the rough ore, and the other with the jigged concentrates and fines); the *ore smelting plant* containing reverberatory furnaces with beds 23 feet 3 inches by 18 feet 9 inches; the *roasting department*, which is equipped with two revolving calcining furnaces, three large roasting furnaces, the beds of which are 20 feet by 13 feet 6 inches, and seven smaller furnaces; one calcining furnace serves the three large roasters, and the second calcining furnace is connected with the smaller ones.

The *refinery department*, which is furnished with three refining furnaces, two of which are kept at work with the third in reserve. The charge of "rough copper" is about 12 tons, and the coal consumed 0.4 ton per ton of refined copper produced. There is nothing of note to mention in connection with the actual furnace operations here, as the work is conducted in the manner usual in a plant of this size, the copper being ladled into the moulds by hand. The only special point to which attention should be called is that the copper cathodes from the electrolytic refinery in connection with the gold and silver department are added to the charge.

The following are the most usual forms in which the refined copper is cast: Ingots, weighing 14 pounds; "medium cakes," weighing about 40 pounds, made specially for the Indian mints at Calcutta and Bombay, and for the Cossipore Shell Foundry; and "cakes" weighing 1 cwt. As a check on the work of the refinery, the chief chemist makes an electric conductivity test of each furnace charge, and unless this is satisfactory, the charge—after ladling—is put on one side and an analysis of the copper is then made to determine whether the defect is due to an excess or deficiency of oxygen, or the presence of too high a proportion of nickel. If the latter is found to be the cause of the low conductivity, the copper is sent back to the roasting furnaces and put through again in small quantities at a time with the other charges. If the examination shows that the fault lies in the quantity of oxygen present, the charge is passed through the refining furnace a second time. It is, however, not often that the copper is thus condemned; the management has prided itself on keeping the quality of the copper as uniform as possible, and any charge which departs from this uniformity is treated in the manner described.

The electrical conductivity test is admirably suited for this work, as it gives a definite numerical result, and with suitable appliances it is easy of execution. During the second or third round of the ladling a special test piece is cast, and this is carefully drawn down by the smith to a short rod about 9 inches long by ¼ inch diameter. One-half of this is then drawn down on a wire-drawing bench, with occasional annealing, finishing through a sapphire die to wire about 0.025 inch diameter. After a final annealing and cleaning the electrical conductivity is carefully determined against a standard coil. The second half of the rod is kept in that form after being labeled and is available for future reference. The electrical conductivity of "Wallaroo" copper is kept at about 88 per cent; if it falls below 86 per cent it is not passed.

The second paper read at this meeting, by Mr. L. A. E. Swinney, dealt with the "Taverner Process," the notes giving in detail the entire working at one of the largest mines in the Rand for the smelting of zinc-gold slimes, from the drying of the slimes to the ultimate production of gold bullion. The figures given pertained to a specified actual smelt from start to finish.

#### MARKET PRICES DURING NOVEMBER.

Prices have ruled very high indeed in regard to the raw materials of chief interest to engineers, owing to the steadily maintained activity of the engineering trades.

Copper sulphate is firm at £31.10 per ton. Shellac is slightly easier at 215s. per cwt., and Para rubber unchanged at 5s. 2d. Platinum is quoted at 150s. per ounce, and seems likely to go higher.

Copper reached £104 about the middle of the month, but closed firm at £102.10 on Nov. 30. Zinc has been in great demand, best sheets fetching £32.15 per ton. Lead, after rising 5s. to £19.17.6 per ton, declined 2s. 6d. to £19.15. Tin has been fairly steady, closing at £197 per ton, cash.

A sudden spurt in iron has raised the price of Cleveland warrants to 62s. 1d. per ton, an increase of 1s. 11d. since Nov. 24, and 4s. 4d. since Nov. 14. Steel rails and plates are naturally firmer.

LONDON, Dec. 5, 1906.

## SYNOPSIS OF PERIODICAL LITERATURE.

## A Summary of Articles Appearing in American and Foreign Periodicals.

## GOLD.

**Cyanide Works Clean-Up Practice.**—The proper equipment and arrangement of cyanide clean-up plants has often not received the consideration it deserves, and in order to bring the subject to discussion, Mr. J. E. Thomas embodied his views in a paper read before the Chemical, Metallurgical and Mining Society of South Africa. The paper is published in the transactions of that Society for December. Mr. Thomas thinks that the clean-up floor should have a separate drainage from the rest of the extractor house floor, so that any washings should not be contaminated with oil and grease from the pumps, and may be run into a sump from which the residue, after settlement, may be easily collected and included in the next clean-up. He also finds it better to have two acid treatment tubs of a certain capacity than one of the same total capacity, as with the two tanks the one may be fed, while the other is threatening to boil over. When there is only one tank, nothing can be done until this has stopped. A stoppage of the acid tanks throws the whole clean-up back and the time lost cannot be made up. In beginning filter-pressing he has always found that it is much more quickly done when the gold slimes are drawn from the bottom of the washing vat than when the suction from the filter press is slung over the side of the vat. As the gold slimes are very heavy, he advises the use of a specially short length of hose for the filter press pump suction. If, when running down the charges from the acid treatment tubs, everything is put through a 64-mesh screen and the contents of the washing vat are kept in agitation while the press is being filled, there is no danger of the rose over the suction drain pipe becoming choked. He also recommends the use of filter papers over the cloths of the filter press, as the filling is thereby facilitated and the life of the cloths is prolonged, no scrubbing being required to clean them. For washing the cakes, he advises the use of hot water, not only on account of its better washing effect, but because the cakes can be got out dry, even if there is only a layer of gold slimes on the papers or cloths. He also thinks it essential that a vat large enough be installed to take all the washes from the washing vat and the filter press, so that the washes may receive further treatment before being run to waste. He states that gold is present in solution in the acid washes, apparently redissolved by hydrocyanic acid. It is, therefore, advisable to have the washes slightly acid, and then to sprinkle zinc fume over the surface, meanwhile keeping the solution agitated. This should be done each time a charge from the acid tubs is transferred for washing. After the clean-up is over, it is advisable to have the contents of the large vat for the washings assayed, after good settlement, and if necessary, more zinc fume can be added before the solution is run to waste. By this means, he claims, the gold in acid washes run to waste may be reduced to 0.07 dwt. per ton, while the ordinary acid washes will probably carry 1.5 dwt. or more per ton. This method is used in all the mines of the Consolidated Gold Fields group. Mr. Thomas also mentions that the motion of the stirring gear for the acid-dissolving tubs should be reversible, as it is often found expedient to reverse the direction of stirring in order to help the dissolving of the zinc and to free any which may have collected around the upright carrying the paddles.

**Treatment of Gold Zinc Slimes.**—A description of Clark's gold refining process, communicated to the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, by Mr. H. T. Brett, Sabiwa Mine, Rhodesia. The slimes may be preliminarily treated with dilute acid, thus removing the bulk of the zinc and other metals with soluble sulphates. If

much calcium carbonate or hydroxide is present, a treatment with dilute hydrochloric acid can be given. The slimes, which need not be thoroughly washed, and should not be dried, are then moistened with a solution containing niter cake dissolved in sulphuric acid. Thus when the mass starts to dry on being heated, the sulphuric acid will become more and more concentrated, oxidizing and attacking the metals. The niter cake enables this operation to be carried out at a higher temperature than could be done with sulphuric acid alone, while the mass also sinters into a cake, which can easily be heated without dusting or loss of gold. The temperature of this operation should be a visible red, and the material when cold should be of a reddish or brownish color without any black patches; the action up to this stage denotes the destruction of the cyanides and ferro-cyanides, sulphides and reducing compounds. Most of the metals have been converted into sulphates and mercury and selenium have been expelled. The next operation aims at collecting the gold together in a form easily dealt with mechanically. While the fused mass is still hot, lumps of clean niter cake are added, preferably in instalments and in such a quantity as preliminary tests have shown to give the best results. The smaller the amount taken the more coherent is the cake of gold and the quicker is the heating over, but the separation may not be so thorough and the washing out of the soluble salts takes longer. If, on inserting an iron rod into the fused mass and withdrawing it, the portion sticking to it cools to a dirty brown mass, the gold has not properly run together, but if the cake on the rod on cooling appears white to greenish, or bluish-white between patches of spongy gold, the action is complete. The fused mass may be poured on a cold iron plate and broken into lumps. For the washing the author prefers to use a box, divided after the manner of the zinc precipitation boxes, a filter cloth below and one in the next compartment. Cake is put into 1 and 3 compartments, hot water trickles into 1, runs through the cake and carries fine suspended particles into 2. These are prevented from escaping by the filter above 2, and so on with the others. The solutions escaping would thus be saturated with silver sulphate when they passed out at the end of the box, and the silver could be precipitated by using a similar box filled with iron or copper or both, say copper in the first and iron in the last. Washing is to be continued until the whole of the silver is removed, there will remain with the gold sand, lead sulphate, some calcium sulphate and sometimes oxide of tellurium and basic sulphate of antimony. In order to avoid production of slags, the author recommends amalgamation of the gold. The sponge gold, free from sulphate of silver, can be readily amalgamated with mercury. The amalgam can be retorted, and if small amounts of impurities were reduced, they are easily gotten rid of by heating the retorted gold in clay pots, adding a small quantity of chlorate of potash and a little common salt. When the latter has melted and run through the porous cake of retorted gold, small pieces of dry, solid niter cake are added, and when the action has ceased, some borax can be poured in, when every trace of base metals will be found to be eliminated and even small quantities of silver removed.

**Treatment of Zinc Box Precipitates.**—The economic treatment of precipitates obtained in the zinc boxes where the gold is precipitated from the cyanide solutions, is a matter of considerable importance. Mr. S. J. Truscott, in a paper before the Institution of Mining and Metallurgy, London, describes the method adopted at the Redjang Lebong Mine, Sumatra, as well as the manipulation of the tilting furnaces in which the



refined precipitate is smelted. The plant contains six zinc boxes, three for the solution from the sands and three for that from the slimes; each box has eight compartments, 36 x 15 inches x 20 inches deep. The precipitation plant is cleaned up twice a month.

One box is cleaned at a time, the solution being displaced by the introduction of water into the top compartment. When the displacement is complete, the zinc in the first four compartments is washed each in its own compartment, and passed over to drain on an adjoining box. A plug in the bottom of each compartment is then opened, and the precipitate, fine zinc and water, pass into the receiving tank, at the entry to which a screen separates the short zinc. The coarser short zinc still remaining in the compartment is then collected in buckets and washed by a water spray in a revolving trommel of 20-mesh screen slanting over the tank. When the first four compartments are finished the washed zinc is returned to them and the remaining compartments are treated in a similar manner. The cyanide solution is then run on again, fresh zinc being added to the lower compartments. The level of the water in the receiving tank is kept down by a small pump, which forces the water through a 20-cell Johnson press, the cells being 24 inches square and 3 inches deep. When all the cells are finished, the solution is pumped out down to the thick slime; the precipitate contained in the press is then returned to the receiving tank. The precipitate in the receiving tank is then treated with acid, the action being aided by a stirrer. When the action is complete, the tank is filled with hot water and the mixture stirred, after which the solution is pumped off through the press. More water is then introduced and the treatment repeated until the water shows no further acidity with litmus paper, after which all the precipitate is drawn into the press and discharged ready for calcination. The short zinc is meanwhile treated similarly with acid in its own tank, and then transferred to the receiving tank for washing as above. The consumption of acid is 0.117 pounds per pound of precipitate, and 0.514 pound per pound of short zinc. The precipitate is calcined in 500-pound lots upon an iron tray over a wood fire at low heat. The material has the following composition: Gold, 8.19 per cent; silver, 58.10 per cent; selenium, 3.56 per cent; zinc, 5.9 per cent; lead, 3.8 per cent; silica, 3.96 per cent; iron, 0.91 per cent; copper, 0.7 per cent;  $MnO$ , 0.12 per cent, and undetermined, 14.76 per cent. With clean precipitate, borax is sufficient for fluxing, the charge being precipitate 100 parts, borax 16 parts. With less clean precipitate the proportion is: Precipitate, 100 parts; borax, 17.8 parts; bicarbonate of soda, 9.0 parts, and sand, 0.7 parts. When it is expected that matte will form scrap iron is added; the latter is invariably the case when smelting the short zinc, and it occurs occasionally with the precipitate. This matte consists mostly of silver selenide, as shown by the following analysis: Gold, 0.51 per cent; silver, 66.86 per cent; selenium, 24.10 per cent; copper, 5.22 per cent; iron, 0.99 per cent; zinc, 0.65 per cent; lead, 0.31 per cent; undetermined, 1.36 per cent. The two tilting furnaces for melting the precipitate have given much satisfaction, and compared with ordinary pot smelting in single furnaces they greatly reduce the amount of manipulation and there is no loss by dusting, spitting or frothing. The furnaces consist of box-shaped castings, provided on two sides with trunnions carried in bearing on iron standards; the tilting is done by worm and wheel gearing. The retort sits at an angle of about  $30^\circ$  upon an arch of fire-brick some inches above the level of the fire-bars, its neck protruding slightly through the front casting of the furnace. The latter is 1 foot 10 inches x 1 foot 10 inches x 3 feet high, the inside dimensions of the retort being: Mouth,  $6\frac{1}{2}$  inches; middle diameter,  $13\frac{3}{4}$  inches; bottom,  $9\frac{1}{2}$  inches; length, 30 inches. The top of the furnace is covered by three loose arches. The initial amount charged into the retorts is 150 to 200 pounds, more being added when this is partly reduced, until about 250 to 300 pounds are reached. This is then com-

pletely reduced, the slag run off into slag pots and the metal poured into molds of about 640-ounce capacity. The fineness of the bullion for a period of six months was 126.86 parts of silver and 818.54 parts of gold, selenium up to 3 per cent being also found, and small amounts of copper, lead, zinc and iron. During the smelting, purple fumes of selenious dioxide, tinged with selenium, are very noticeable, in such quantity that much is deposited in a pipe generally kept at the mouth of the retort for the purpose of drawing off these fumes.

**Sand Sampling in Cyanide Works.**—The important question of the most reliable method for taking samples of treated and untreated sand in cyanide plants has been investigated by Mr. D. Simpson, at the New Goch Gold Mining Co's mill on the Rand. He communicates his results in a recent paper before the British Institution of Mining and Metallurgy. The plant in question contains sixteen vats arranged in two lower and two upper rows; they measure 40 feet in diameter by 7 feet 6 inches deep to the filter mats. The upper settling tanks are filled by means of hose delivering the pulp from a central distributor, while the sand is afterwards transferred to the bottom tanks by shoveling through seven doors. The first method investigated was the taking of samples by the so-called sampling rod, a wrought iron pipe 7 feet 6 inches long by 2 inches internal diameter, slit along one side to within 6 inches of the T-handle. Samples were taken haphazard over the surface of the settled sand. The author states that assay results from the samples thus obtained were so irregular and varied to such an extent when taken from the same charge, that this method is unreliable. In the second method samples were withdrawn at points nine in all, with the sampling rod, at equal distances apart, along a diameter. The contents of a 40-foot tank were divided into four parts by weight, by describing on the surface circles with radii of 10 feet, 14.14 feet, 17.32 feet and 20 feet. These divisions should be represented by equal fractions of a proper sample, but marking off nine points an equal distance, 4 feet apart, five are found to be within the inner circle, two within the next, two more within the third, and none at all in the remaining outside circle. Thus seven parts by weight of the sample represent half the contents of the tank and only two parts the remaining half. The author obtained consistent results by this method, but they were consistently high. The third method was to describe the four circles as before, and a line was then drawn across the center of the tank at right-angles to the diameter along which the sampling was to be done. Thus the sand is divided into eight equal parts by weight. Each portion of the sample must then be taken from what may be called the center of mass of each section, namely, in the 40-foot tank a tubeful of sand must be withdrawn at points situated 7.07 feet, 12.2 feet, 15.81 feet and 18.7 feet distant from the center on each side along the diameter, care being taken to insert the rod perpendicularly. This last method, in the author's opinion, is the most trustworthy way in which to take rod samples, as any occasional variation from working accuracy has been, in each case, traced to bad manipulation of the delivery hose. In carrying out the fourth method, half the sand was discharged from the tanks, leaving at the diameter a vertical face 7 feet high, which was then divided into seven horizontal zones, each 1 foot wide. The face of each zone was scraped into a bucket, and the seven samples were treated individually and collectively. The fifth method consisted in dividing the seven cones formed in the bottom tank underneath the doors of the upper tank, at a certain period during the discharge of the upper tank into quadrants and sampling them with a 2-foot rod. The sixth method was carried out by stationing a man in the lower tank, who caught with a spoon what he could, while the material fell down. A seventh method for the discharged material consisted in dropping the heated sand on three conveyor belts, which delivered to a single belt running at right angles. During each complete revolution of the belt a man took a small scoopful of sand as a mark on the belt passed him, and continued doing

this until the tank was discharged. This method is now carried out regularly at the plant for taking discharge samples. In the eighth method a sample was taken from each tram car, into which the sand delivered by the conveyor was drawn from the bins. The author states that the assays obtained by the latter method were usually the same as those by method seven. The author advises the managers of sand plants, that the occasional resort to such a method as No. 4, and the systematic compilation of the results in tabular form, will repay many times over for the trouble entailed in the value of the index to the treatment obtained.

#### SILVER.

**The Present State of the Metallurgy of Purely Silver Ores.**—In a paper read before the recent meeting of the American Mining Congress, published in the *Canadian Mining Review*, December, the author, Mr. J. Wilding, intends to call attention to the fact that the metallurgy of purely silver ores has been very much neglected of late years. He reviews briefly the processes for the extraction of the silver from the ores at the place of their production. The amalgamation of raw ores in pans has almost gone out of use, owing to the exhaustion of suitable oxidized ores. The chlorination of the ore, followed by pan amalgamation has the disadvantages of high losses and in most localities of high costs, the losses being partly due to imperfect chlorination and partly to the volatilization of chloride of silver in roasting, which is never less than 7 per cent, and may reach 30 per cent of the silver contents of the ore. Chlorination, followed by leaching with hyposulphite has the same losses as the previous method, and in addition the sulphides obtained have to be further treated, the latter operation, however, being usually compensated by the lower cost of leaching. The Russell process is stated by the author to have generally failed to give satisfactory results with raw ores, and can only be regarded as an aid to the older hyposulphite process in cases of imperfect chlorination. The Patio process still renders good service in a few places in Mexico, being in Pachnea usually preceded and often followed by concentrating out the sulphides and a large part of the small amount of gold contained in the ore. An extraction of as much as 90 per cent of the values is claimed on ore containing 1,500 grams of ore per metric ton, though the proportion extracted from ore of half this grade is much less. Recent improvements introduced into the process are the finer grinding of the ore and the introduction of mechanical devices for turning the ore over in the torta. As far as the use of potassium cyanide as a solvent is concerned, the author states that in cases in which a sufficient proportion of the silver is associated with sulphides removable by concentration, fair metallurgical results have been obtained, even though the proportion of the silver content of the concentration tailings extracted by the cyanide has not exceeded 50 to 60 per cent. As the cost of treatment is not high, the author looks for a further extension of this method for such ores. He closes with a plea to mine owners not to neglect metallurgical research in the treatment of low-grade silver ores.

#### IRON AND STEEL.

**Magnetic Preparation of Ores.**—C. Blömeke, of Aachen, describes in *Metallurgie* of Nov. 8, a new system used by the Hernalthaler Ungarische Eisenindustrie-Aktien-Gesellschaft, in Budapest. The separator is a magnetized steel roll, revolving fast, onto the upper surface of which the material is charged. The speed is so high that centrifugal force plays a considerable part in the separation. On top of the roll non-magnetic material is thrown off tangentially, magnetic material adheres. On the side of the roll, gravity aids centrifugal force in drawing off the material tangentially, and feebly magnetic materials drop off under the combined pull of these two forces. At the lower part of the roll gravity commences to act against centrifugal force, causing particles on the rising side to adhere more strongly; such are removed by a revolving brush, on the

ascending side of the roll. A modification of the apparatus, adapted for moist or wet material, has a more slowly revolving, with its lower side immersed in a tank of water, into which the non-magnetic particles will drop as they pass through, while the magnetic particles adhere and are brushed off outside the water bath. Tin ores containing wolframite, similarly treated, had 96.3 per cent of their wolfram removed, saving at the same time 95.6 per cent of the tin.

**Graphite Formation.**—F. Wüst, in *Metallurgie* for Nov. 22, discusses the formation of graphite in cast iron. A series of eight tests, in which the iron stayed liquid 70 to 350 seconds at the eutectic point,  $1,130^{\circ}$ , showed that the amount of graphite formed is dependent upon the velocity with which the iron passes through this halting point. Those other elements, such as silicon, which are known to increase the tendency to form graphite, act by prolonging this setting interval; on the contrary, manganese and sulphur shorten the time of setting, and therefore decrease the formation of graphite. According to this new view it is quite possible to neutralize the effects of such elements as silicon by rapid cooling through the setting point, and of such elements as manganese by slow cooling at this temperature. This was proved by casting an iron high in manganese and low in silicon in a clay mould, which was kept at bright redness, resulting in producing a casting containing 60 to nearly 80 per cent of its carbon in the form of graphite, that is, a completely grey iron, from a material which under normal casting conditions was completely white, and contained less than 5 per cent of its total carbon in the form of graphite. These investigations are of the greatest interest and importance to foundry practice.

**The Art of Cutting Metals.**—President W. L. Taylor, of the American Society of Mechanical Engineers, delivered an address before that society in New York on Dec. 4, which deserves more than passing comment. The address, as printed, is an epoch-making treatise on this subject, recording with great clearness the results of twenty-six years systematic research in shop and laboratory. The plan of the work was to determine the effect of each of twelve variables entering into the problem, which are stated and discussed as (a) the quality of the metal which is to be cut; (b) the diameter of the work; (c) the depth of the cut; (d) the thickness of the shaving; (e) the elasticity of the work and of the tool; (f) the shape or contour of the cutting edge of the tool, together with its clearance lip and angles; (g) the chemical composition of the steel from which the tool is made, and the heat treatment of the tool; (h) whether a copious stream of water or other cooling medium is used on the tool; (j) the duration of the cut, i. e., the time which a tool must last under pressure of the shaving without being reground; (k) the pressure of the chip or shaving upon the tool; (l) the changes of speed and feed possible in the lathe; (m) the pulling and feeding power of the lathe. The determination of the single effect of each of these twelve variables, keeping the other eleven constant, was a problem requiring the highest technical skill and patience. The practical results attained are classified under four general heads: (A) The codification of the facts or laws connected with the art of cutting metals; (B) the statement of mathematical expressions of those laws simple enough to be suited for daily use; (C) the limitations and possibilities of metal-cutting machines; (D) the development of a slide rule which embodies the application of these laws and expressions to the work of the machinist, enabling him to fix quickly upon the best conditions for each kind of work. The net result of all this work is to take the control out of the hands of the workman and put it into that of the management, thus superseding "rule of thumb" by scientific control.

**Etching Fluid for Metallographic Work.**—Mr. Kourbatoff, in the *Revue de Metallurgie*, November, states that after visiting the principal metallographic laboratories at the French steel works, he has found that a solution of 4 per cent nitric

acid in isoamyl alcohol is generally employed. The reagent, however, which serves to distinguish troostite and sorbitic matters is not employed, as it is claimed to give bad results. The composition of this reagent is 1 part of a solution of 4 per cent nitric acid in acetic anhydride, 1 part of methyl alcohol, 1 part of ethyl alcohol and 1 part of isoamyl alcohol. The author claims that the cause of the bad results referred to is due to the use of the acetic acid, properly so-called, instead of its anhydride ( $C_2H_5O$ )<sub>2</sub>O. It is indispensable to employ exclusively the anhydride. Moreover, care has to be taken that the solution is not prepared too much in advance so that the formation of ether is obviated. In order to succeed with the reagent the following precautions have to be taken: Two separate solutions are first prepared, namely, 1, solution of nitric acid with a concentration of 4 per cent, in acetic anhydride; 2, a mixture of equal parts of methyl, ethyl and isoamyl alcohol. At the moment of using the liquid, one part of solution 1 is added to three parts of solution 2. By using nitric acid of 1.32 density the reagent does not produce for the first 10 minutes any coloration on materials other than sorbite and troostite. On the other constituents which remain colorless one observes only some contour of crystals.

**Blast Furnace Gases.**—In a communication to the *London Engineering*, Nov. 30, Mr. Marcus Ruthenburg proposes an explanation of the indifferent success obtained in the United States in the application of blast furnace gases to the driving of large gas engines. Furnace practice differs materially in the United States and Europe in regard to the temperature of the blast, the ore analysis and the speed of driving, and consequently also in the quality of the effluent gases. He cites the following experience with a Westinghouse three-cylinder engine, with cylinders 15 inches in diameter by 22-inch stroke, running on blast furnace gas of an average of 100 B. T. U. per cubic foot. The blast furnace gas had been passed through the ordinary downcomers and dust catchers, and then through four washers and scrubbers and a meter, before entering the engine. A sample of the gas taken next to the engine at any time was practically free from suspended matter. The engine after running for three weeks continuously was taken down to insert new rings and dress the cylinders. It was then found that the ports, which were normally  $1\frac{1}{4}$  inches wide, had narrowed down to little more than  $\frac{3}{8}$  inch, on account of a glassy incrustation, which had to be chipped out with a hammer. Upon analysis this incrustation was found to contain every element that had been in the blast furnace charge. As the gas did not show any suspended matter, the author accounts for the formation of this deposit by assuming that the various elements of the furnace charge were dissolved in CO as carbonyls, which carbonyls, being true solutions in CO, and therefore not suspended matter, were not to be removed by filtering or washing. Upon explosion, in which the carbonyls took part, the metals were deposited as oxides in the glassy scale mentioned above.

#### COPPER.

**Pyritic Smelting.**—R. Sticht, in *Metallurgie* for Nov. 8, calls attention to the fact that the iron pyrites charged into the pyritic smelting furnace not only loses half its sulphur by volatilization before being oxidized, as is generally assumed, but that his tests show an even lower sulphide than FeS, possibly Fe<sup>2</sup>S, to be present at points in the furnace where no free oxygen exists. The conclusion drawn is that more heat is evolved in the furnace by the oxidation of iron than from oxidation of sulphur, instead of the reverse being the case. The balance in favor of the iron is even more than Sticht concludes, because he omits to add in the heat of combination of the iron oxide with silica to form slag, which is by no means a negligible quantity.

#### LEAD.

**Condensation of Fume.**—K. Friedrich, in *Metallurgie* for Nov. 22 and 29, reviews this whole subject. The two most

important factors are cooling of the gas and contact with large surface. For attaining both objects simultaneously there is suggested the use of two chambers similar to Siemen's regenerators. The warm gases pass through one chamber, heating it and depositing fume on its surfaces, while the companion chamber is being cooled by the cold air passing on its way to the furnace; when the first chamber is warmed to its further end, the current is diverted to the second cool chamber, while the first is cooled down in its turn by the cold air. The bricks cannot be in form of checker work, because of the difficulty of cleaning; they are built similarly to hot-blast stoves, with straight passages occupying three-quarters of the available space. Cleaning off is accomplished by water spray running through these channels, in between two reversals. A modification of this idea is to use a number of such chambers, say ten, arranged circularly like the compartments of a Hoffman brick kiln. The warm current passes, let us say, through 6, 7, 8, 9 and 10, while the cooling current passes through 1, 2, 3, 4 and 5. When 10 becomes too warm the valves are shifted so that the hot current passes through 7, 8, 9, 10 and 1, while the cooling current goes through 2, 3, 4, 5 and 6. By continuing this change, always in the same direction, a very effective cooling and fume deposition is accomplished.

#### CARBIDES.

**Decomposition by Reagents.**—C. Hahn and A. Strutz describe in *Metallurgie* of Nov. 8, experiments on the decomposability of carbides. Calcium carbide heated to 480° C. and treated by dry steam formed hydrogen, methane and heavy hydrocarbons, leaving a deep black residue, which treated by hydrochloric acid furnished carbon of a peculiar kind. Dry hydrochloric acid gas at 400°, furnished a gas consisting principally of hydrogen, with a small amount of heavy hydrocarbons. The melted residue, consisting principally of calcium chloride, furnished also pure carbon on crushing and washing. Dry hydrogen sulphide furnished gas which was almost entirely hydrogen, and the residue, treated by hydrochloric acid, furnished a similar kind of deep-black carbon. Aluminium carbide acted similarly to calcium carbide. Manganese carbide, acted upon by water acidulated by hydrochloric acid, furnished almost pure hydrogen; dry hydrochloric acid gas gave a similar gas. The residue gave carbon having a luster like graphite, entirely different from the carbon obtained from the other carbides. Silicon carbide (carborundum), treated by dry water vapor at 1,300° to 1,400°, gave hydrogen gas with some carbon dioxide similar to a water gas process. The separation of pure carbon under these circumstances was not proven.

#### PYROMETRY.

**Thermo-Elements.**—Excellent work is being done in Washington on the subject of pyrometry, notably by the Bureau of Standards and the Geophysical Laboratory of the Carnegie Institution. From the latter a paper by W. P. White has emanated, which is published in the December issue of the *Physical Review*, and in which the author shows that the thermo-element may be made an instrument of highest precision. The main source of error is inhomogeneity. The effect of the initial chemical inhomogeneity is practically negligible, while the effect of physical inhomogeneity—that is the variation in hardness—can by no means be neglected. New wires are usually hard drawn. This hardness as well as that which results from bending, etc., is removed from platinum wires by annealing. On constantan and copper wires a film of oxide or other tarnish is apt to form, which must be cleaned off to restore the original calibration. At high temperatures platinum takes up impurities which diffuse into the wire. Iron and silicon are perhaps the most important. In an oxidizing atmosphere contamination occurs above 900° from the presence of small quantities of rhodium, and especially iridium, either pure or in alloys with platinum. The contaminating impurity cannot be removed, but can be excluded by enclosure in glazed Marquardt porcelain.



## MISCELLANEOUS.

**Turbo-Blowing Engines.**—In a paper before the West of Scotland Iron and Steel Institute, and published in the *Journal* of that Society, October, 1906, Mr. H. M. Brown gives some data concerning the performances of turbo-blowing engines. Engines of that type consist of a turbine air compressor directly coupled to a steam turbine. The latter is of the usual type, and the air turbine consists of a cast iron cylinder in which is arranged a concentric shaft mounted on bearings and revolving at high speed. The annular space between the cylinder and the shaft is occupied by rows of alternate guide and moving blades, the guide blades being arranged on the cylinder and the moving blades on the shaft. Air, on entering the turbine, meets the first row of moving blades, after which it is directed by the first row of guides into the second row of moving blades, and so on, until it reaches the required pressure at the outlet. The efficiency of these machines is very high and the delivery of the blast is steady and continuous. By arranging a by-pass on the steam turbine, by means of which high-pressure steam is admitted to the low pressure of the turbine, the pressure and volume of the blast may be temporarily increased to 40 to 50 per cent above normal, so that a furnace can be blown free if it should be choked. The governing of the turbo-blowing engine is effected by a sensitive centrifugal governor, which actuates the steam admission valves by means of a steam relay. The usual size of the engine is about 20,000 cubic feet of free air per minute, against 10 pounds normal blast pressure when running at 3,000 r. p. m. With the by-pass mentioned above, the speed can be raised to 3,800 r. p. m., and the output increased 50 to 60 per cent, so that either a higher pressure can be obtained or a greater volume of blast. The author states that on account of the steady and continuous blast obtained from such a machine, smelting furnaces are found to be much less liable to hanging or scaffolding. The weight of the apparatus is only 25 tons, as compared with 140 tons for a reciprocating blowing plant. He quotes the performance of a turbo-engine at a large smelting works at Middlesbrough, England, as a typical case. This engine was delivering 18,500 cubic feet of free air per minute, against 10 pounds blast pressure, and took 15,100 pounds of steam per hour. The turbine was fed with saturated steam at 150 pounds pressure, and exhausted into a 27-inch vacuum. The steam consumption was, therefore, only 22.5 pounds per adiabatic horse-power-hour, or an over-all thermal efficiency of 37.4 per cent. Another type of machine is arranged for dealing with large volumes at low pressures, and is used for gases from blast furnace plants, pumping and forcing them through the recovery plant. The exhaustor is directly coupled to a steam turbine, and its shaft carries three screw propellers, arranged to run at any speed up to 8,000 r. p. m. Between the propellers are arranged stationary guide blades. When higher water gauges than 28 or 30 inches are required, it is only necessary to add to the number of propellers. The author remarks upon the noteworthy point, that these machines, when dealing with blast furnace gases, do not become clogged with tar or other matter, owing to the high velocity of the gas through the machine.

**Treatment of Arsenic and Antimony Ores.**—A note in the *Mining Journal*, London, Dec. 1, translated from the *Echo des Mines*, Nov. 15, treats with experiments on ores from the Lucette, by the Herrenscheidt process, according to the wet process patent by the latter. The fine antimony ores or the residue from old liquation furnaces are placed in an autoclave with carbonate of soda and lime, or sulphide of sodium or sulphur, carbonate of sulphur (?) and lime, according to the colors intended to be obtained in the finished product. The autoclave is furnished with a plunger, and when the pressure is 5 atmospheres it is emptied by the plunger into settling tanks. The clear liquor passes into coke chambers, as also the sulphuric acid from the roasting furnaces for the antimony ore or the oxide furnaces. Thus the antimony is precipitated now

through the filter presses and dried. It is then smelted for regulus or sold for the manufacture of india rubber or colors. The residue is stated to contain the gold which is associated with the antimony. The experiments are stated to have been made on an industrial scale with several tons of ore.

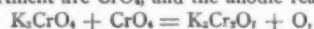
## ANALYSIS OF CURRENT ELECTRO-CHEMICAL PATENTS.

**Dental Furnace.**—F. E. Roach, 838,647, Dec. 18. Application filed Oct. 11, 1905.

Mechanical details of construction of a dental furnace. A muffle with resistance wires in its walls is placed in the center of a furnace, the free space between the exterior furnace walls and the muffle being filled with asbestos. The article (porcelain, etc.) to be baked is placed on a support in the center of the muffle and means are provided for subjecting the article to more or less heat.

**Converting Chromates into Bichromates.**—Robert Suchy, 838,757, Dec. 18. Application filed July 9, 1903. Assigned to Chemische Fabrik Griesheim Electron.

A saturated neutral solution of alkali chromate is introduced into the anode and cathode compartments of an electrolytic cell, separated from each other by a diaphragm, employing as electrodes an anode not attacked by the solution (like lead or platinum), and "a cathode which will not reduce the chromate." When electrolysis is started the only anions in the anode compartment are  $\text{CrO}_4$ , and the anodic reaction is



that is, formation of bichromate and evolution of oxygen, while in the cathode compartment the K cations form caustic soda with evolution of hydrogen. To keep the voltage low, it is advisable to heat the bath to 80° or 90° C.

**Electrotyping.**—A. Gerstner, 838,977, Dec. 18. Application filed April 20, 1906.

In electrotyping practice it is usual to submit the wax model, after it has been built up, to the leading process, then polishing the mold, then submitting the mold to the action of a blower for blowing out the superfluous particles of graphite, then submitting the mold to a washing process, then submitting the mold to a treatment consisting in dipping the mold in a bath of copper sulphate, sprinkling iron filings on the mold, and subsequently washing the face of the mold with copper sulphate. All these steps have been usually taken before the mold is stopped off or trimmed and submitted to the action of the battery for electroplating the same. The present inventor endeavors to simplify the process by cutting out a number of the above steps. The only step which is performed after the graphite has been blown out subsequent to the building-up step, is to apply to the mold a solution composed of nitrate of silver (1 ounce), chloride of sodium (2 ounces), and graphite (2 pounds) in water (1 liter). This solution is applied by means of a sprinkling hose attached to a rotary pump connected with a tank containing the solution. The mold is held over the tank, and the solution flows back into the tank after the mold has been treated. The mold is then directly submitted to the electroplating bath, when the coating of silver chloride "will combine with the copper in the electroplating bath to form a metallic silver coating."

**Gold Plating.**—W. S. Hutchinson, 838,716 and 838,717, Dec. 18. Application filed July 29, 1905.

When an article has been electroplated with gold it is necessary to remove the adhering solution by rinsing in water. The object of this patent is to recover the gold from the washing water. The plating tanks are arranged in two rows, and between them a series of rinsing tanks is provided, and below the latter a series of zinc boxes. Washing water is

passed through the series of rinsing tanks from one end to the other, while the plated articles are rinsed successively in each of the different rinsing tanks, beginning with the rinsing tank furthest away from the entrance of the washing water, and therefore containing the most concentrated cyanide solution. From this last rinsing tank the solution enters the first zinc box just below and then passes through the different zinc boxes, the gold being there deposited and the water, freed from the gold, being used over again as wash water. No. 838,716 refers to the apparatus, 838,717 to the process.

**Metallizing Wallpaper.**—E. L. Livingston, 838,346, Dec. 11. Application filed Jan. 25, 1906.

The wall paper which is to be metallized on one side only is wetted and laid on a vacuum board. This board is provided with a series of grooves, which are connected by means of pipes with an air pump, so as to produce a partial vacuum. The paper is thus sucked against the board and held flat. Its exposed face is then covered with an adhesive material, such as show enamel or asphalt varnish and a graphite or metal powder, such as copper bronze, is then applied by a brush. Contact wires are then extended along the longitudinal opposite edges of the paper, and the board and paper are then lowered into the plating bath.

**Cleaning Storage Batteries.**—J. W. Aylsworth, 837,773, Dec. 4. Application filed Sept. 8, 1904.

The object is to remove the very thin and almost invisible films of grease from the interior of the Edison battery (which grease films cause excessive foaming during the charging of the cells). This is done by filling the battery with a weak alkaline solution (about 2 per cent) and maintaining it for several hours at a temperature slightly below the boiling point.

**Charging Storage Cells.**—I. S. Raymer, 838,583, Dec. 18. Application filed Aug. 23, 1905.

A number of storage cells are sub-divided into various groups, and by means of a special commutator, which is the subject of this patent, it is possible to connect and disconnect, successively, each of the groups of cells to and from their charging and discharging circuits.

**Fixation of Atmospheric Nitrogen.**—K. Birkeland and S. Eyde, 837,277, Dec. 4. Application filed Nov. 10, 1904.

The patent refers to a modification of their well-known process of fixation of atmospheric nitrogen by arc discharges (see, for instance, our Vol. II., p. 399, and Vol. IV., p. 126). If it is attempted to increase the tension at the arc for the



FIG. 1.—ELECTRODE CONSTRUCTION.

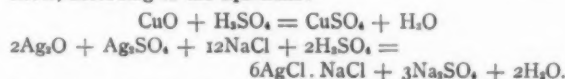
purpose of obtaining an increase of energy as well as arcs with increased surface by increasing the relative distance between the electrodes, the arc will soon be extinguished; i. e., a certain current and magnetic field corresponds to a certain maximum distance between the electrodes. Experience shows that the extinction will always take place when the current is about zero. The object of the inventors' arrangement is therefore to avoid extinction at this point, whereby it becomes possible to increase the tension materially without having to fear an extinction of the arc. Three different methods for obtaining this result are described. They are distinguished as the mechanical, magnetic and electric methods. The "mechanical" method uses one fixed electrode, while the other one *b* can rotate around its axle, as indicated in the upper diagram of Fig. 1. The latter is rotated in synchronism with the alternating current used, in such a way that the distance between the electrode points is a minimum for zero current. The lower diagram shows the use of tubular electrodes, which are especially useful for leading the gases into the furnace through the interior of the hollow electrodes, whereby the latter are cooled. The

"magnetic" method employs a pulsating or variable magnetic field, "such pulsation having an effect on the factor of energy substantially equivalent with the variation of the distance between the electrodes." Finally, the "electric" method makes use of an auxiliary circuit, which acts as a bridge for the arc of the working current.

## RECENT METALLURGICAL PATENTS.

### COPPER.

**Slimes Treatment.**—H. N. Thomson and F. Laist (832,176, Oct. 2) patent the following treatment of slimes derived from ore-dressing apparatus operating on crushed copper sulfide ores not previously roasted. These slimes carry more or less silver and some iron, all the metals being in the form of sulfides. The slimes are dried, disintegrated and roasted at a low dull-red heat. The sulfides are thereby changed partly into oxides and partly into sulfates. The roasted slimes are then agitated in a solution of sulphuric acid (2 to 6 parts by weight), ferrous sulfate (5 to 25 parts), and sodium chloride (5 to 25 parts). The copper oxide is thereby changed into sulfate, while the sodium chloride acts as a solvent for the silver, according to the equations:



To facilitate the solution of the copper during the agitation, air and sulfur-dioxide gases from roasting furnaces are passed through the solution. After filtration the solution is subjected to sulfureted-hydrogen gas, throwing down insoluble copper sulfide and silver sulfide, which are then smelted. The sulphureted hydrogen is obtained from treating a low-grade matte with sulphuric acid.

**Converting Matte.**—In bessemerizing copper matte in presence of silica, E. H. Hamilton (837,562, Dec. 4) recommends the additions of suitable compounds of the alkaline earths (quicklime, limestone, sulfate of lime or barium compounds) to the molten matte at the time of charging or immediately afterwards and during the blowing of the charge. The effect is said to be in a measure a mechanical one, every available particle of iron sulphids being brought into intimate and quick contact with the silica flux and slagged. This mechanical effect is due to the evolution of gases from the alkaline-earth compounds—such as carbonic acid where lime is used—which have the effect of supplementing the agitating and stirring action of the blast.

### ZINC.

**Roasting for Wet Process.**—A patent of Geo. O. Angell (837,273, Dec. 4) is interesting, since it contains the first authentic information on a new process which is stated to be tried on a rather large experimental scale in a plant in New Jersey. The process is understood to involve leaching and electrolysis, but the chief element of interest is the previous roasting process, which alone is the subject of the present patent. The object is to produce soluble sulfates direct from zinciferous ores which are sulfide blende. The chief endeavor is to prevent the sulfates from being broken up into oxides after the sulfate-forming stage in the treatment has been reached. For this purpose the roasting furnace is closed after the roasting has proceeded so far as to cause the ore to give off and ignite its constituent sulfur gases. Then by admission of air and steam into the roasting chamber the ore sulfides are caused to be sulfated by the sulfur dioxide and trioxide primarily obtained.

**Retort Construction.**—A. L. J. Queneau, of the New Jersey Zinc Co. (837,883, Dec. 4), describes the construction of a composite metallurgical vessel. The wall is made up of finely-communited graphite, suitably mixed with fire-clay and sand

to form the desired wad. But the gaseous products of combustion would tend to burn the graphite out, leaving the outer surface in a pitted or porous condition. For this reason the vessel is provided with a thick exterior protective layer of fire-clay and a thinner interior layer of the same material. The function of the latter is to protect the interior of the vessel during the kiln-tempering operation.

#### STEEL.

**Steel-Welding Compound.**—G. Wiesmann (837,443, Dec. 4) patents the following compound for welding steel. It consists of 1 ounce saltpeter, 6 ounces borax, 4 ounces brick or marble dust, and 6 ounces of iron scales. When steel has been burned through, being brought to more than a cherry heat, it is said it can be welded by using this compound equally as well as unburned steel.

#### NICKEL.

**Orford Process.**—Robert R. Maffett, of the International Nickel Co., recommends to modify the Orford process for nickel and copper separation by using a polysulfide of sodium instead of simple sulfide of sodium. The effect is a great reduction in the number of necessary smeltings. The polysulfide is simply produced by adding to the usual charge of sodium sulfide in the cupola furnace 3 or 4 per cent of its weight in crude sulfur.

#### METAL PRECIPITATION.

**Metal-Depositing Apparatus.**—W. A. Hendryx (837,832, Dec. 4) uses the apparatus shown in Fig. 1. The solution in which the metallic values are dissolved is supplied through pipe 15 to the revolving cylinder 5, which contains the precipitating agent 24 in granular form. For the deposition of copper from its chloride or sulphate, iron or steel scraps or

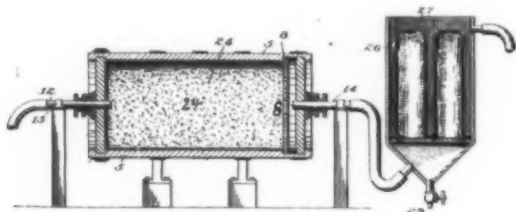


FIG. 1.—METAL DEPOSITION.

shavings are used as filling material, while for cyanide solutions of gold and silver metallic zinc is used in shot or pulverulent form. Near the end of the cylinder the screen 8 is used with apertures of such size as to retain the granular depositing material 24, while permitting the free passage of the solutions carrying the precipitated metallic values in suspension. The outlet 14 is connected with the lower end of the receptacle 26, containing one or more filters 27. The precipitated values are collected in the lower portion of the tank 26, to be withdrawn through outlet 29.

## BOOK REVIEWS.

**A LABORATORY GUIDE TO THE STUDY OF QUALITATIVE ANALYSIS**—based upon the application of the theory of electrolytic dissociation and the law of mass action. By E. H. S. Bailey and Hamilton P. Cady. Fifth edition; thoroughly revised. 12mo., pp. 278. Philadelphia, P. Blakiston's Son & Co.

It is a great pity that a very good book on qualitative analysis, by experienced teachers, should have been spoiled as this has been by the introduction of scientific gibberish which all the skill of the writers cannot make either clear or useful. The standpoint of the author is clearly expressed in the title, and they have endeavored conscientiously to carry it through,

but at the cost of immense inconsistency and almost total destruction of the usefulness of their work as a guide to scientific instruction.

The whole scheme and tendency of the book can be none other than to lead the student away from the simple facts of qualitative analysis, and to plunge him *ab initio* over head and ears into a slough of scientific theory. Subjected to such treatment the unfortunate beginner can develop no independence of thought, no fresh incentive to observation; he is simply tied down to expressing himself in the shibboleth of "a creed outworn."

One of the most insidious tendencies of modern instruction is that of bringing up the young in the narrow lines of a creed or theory. The religious zealot exclaims, "Let me have the children from 6 to 16, and I care not who has them the rest of their lives." Intellectually, children thus brought up are like Japanese stunted trees. The scientific enthusiast says, "Let us instruct the beginner, from the beginning, in our theoretical speculations, and as he grows up he will talk our language and see nature's facts only from our standpoint." Nothing can destroy the freshness of approach, the originality of thought, quicker than these insidious tactics. If we want to develop strong, original, manly investigators, afraid of no fact, because it is nature's fact, and equip them to master natural phenomena, we must teach them from all standpoints; they must *ab initio* be taught to be eclectics.

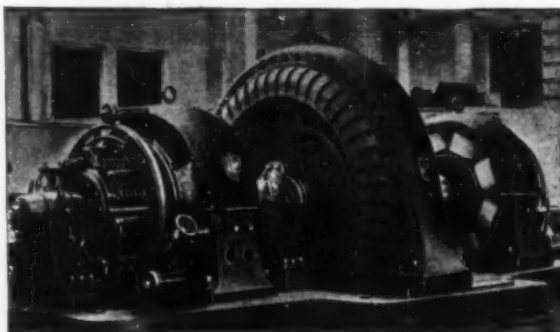
We earnestly appeal to all true teachers, anxious and striving to develop in their students independence of observation, originality of thought, freedom from narrow thinking and broadness of view, to avoid the use of this book—or, perhaps, if a copy finds its way into their hands, treat it as setting forth most clearly how qualitative analysis should *not* be taught.

JOS. W. RICHARDS.

#### Motor Generator.

The adjoining illustration shows one of three motor generator sets built by the Allis-Chalmers Co., Milwaukee, for the Niagara Electrochemical Co., at Niagara Falls. The electrolytic production of metallic sodium from fused sodium hydroxide must be carried on continuously and a complete shut-down should never occur at any time.

For this reason the set consists of one motor and two gen-



ELECTROLYTIC MOTOR GENERATOR.

erators, and either generator can be disconnected in case repairs are necessary without interfering with the operation of the remaining machine.

The motor is of the synchronous type operated with 25-cycle alternating current at 2,200 volts, the speed being 500 r. p. m. On each end of the motor there is driven by a flange coupling a 200-kw. direct-current generator, the e. m. f. of which is capable of variation through a considerable range, the maximum value being 165 volts.



### Automatic Water Still.

Distilled water is a household necessity for drinking purposes in many towns, but it is no less a necessity for constant use in many industrial works. The new water stills described below, made by the Wagner Water Still Company, 87 Washington Street, Chicago, are specially interesting on account of their absolutely automatic operation, since they require no care or attention, operating continuously when the water and gas or steam supply are turned on.

The smaller stills are made in four sizes, distilling from  $\frac{1}{2}$  gallon to 2 gallons per hour, and are operated by gas; they are made of nickel-plated copper. The larger stills are operated by steam, and are built for capacities of from 100 to 1,000 gallons daily. They are designed especially for the use of large laboratories, for filling storage batteries, etc. Both types of stills deliver the distilled water cold, aerated and chemically pure.

Fig. 1 shows the small type, for household purposes and for the use of chemists, assayers and smaller laboratories. The still consists of a boiler G and a condenser tube, shown at the right-hand of the illustration, both being connected by a high goose neck H—which prevents any physical matter from passing over with the steam under any conditions. C is the water connection while the gas connection is at K, the gas being supplied to the Bunsen burner B.

The construction provides for a continuous in-flow of water at C. When it rises to A the boiler G receives water up to a point level with the overflow L. When the water evaporates in the boiler more water is supplied through the inlet A, and a continuous level is thus maintained in the boiler. The cock D is for draining the condenser casing and boiler when the apparatus is not in use.

The boiler chamber is instantly detachable by means of the slip joints P and N and the thumb screw at A for cleaning purposes.

In action the water flows through the cock C upwards, and condenses the steam in the tube H, supplying the boiler as indicated and the surplus passing off through the overflow F in a small hot stream. The distilled water drops from the nozzle E at the bottom chemically pure, cold and aerated.

The larger size of the Wagner automatic water still is shown in Fig. 2, the principle being the same as for the smaller type. This larger size is operated with steam, the steam entering through the valve A and leaving through the valve B. The bottom C of the boiler D may be unscrewed for cleaning purposes by means of two detachable brass pins EE. The pins are

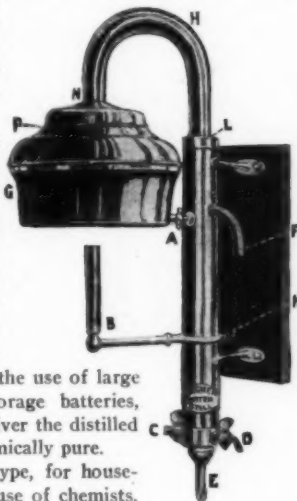


FIG. 1.—WATER STILL OPERATED BY GAS.

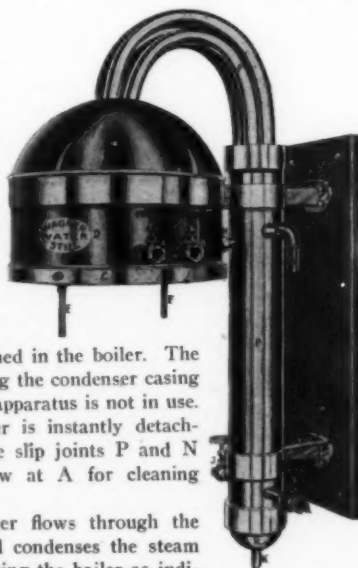


FIG. 2.—WATER STILL OPERATED BY STEAM.

made detachable so as to prevent the still from being tampered with.

The water enters through the inlet valve J. When it rises to H the boiler is supplied with water to a point level with the overflow G. As the water evaporates in the boiler D more water is supplied through H, thus maintaining a continuous level in the boiler. As the steam passes through the tubes F it is condensed in the condensing chamber P and delivered cold, aerated and chemically pure from the nipple K. R is the collecting cup from the ends of the steam tubes L. All interior parts of the apparatus coming in contact with the steam or condensing water are heavily lined with pure block tin, so that an absolutely pure distillate is delivered. All parts are of heavy polished copper and brass.

The steam coils in the boiling chamber are designed to stand 150 pounds steam pressure, but a pressure of 35 to 60 pounds—or 40 pounds in the average—is sufficient. The water supply through the valve J is so regulated that a small hot stream is delivered through the overflow G, which can be led to a heater tank and used for other purposes.

It is stated that distilled water from this still is furnished at a maximum cost of  $\frac{1}{2}$  cent per gallon, and that this cost is very appreciably reduced when the overflow water is used.

### Notes.

**American Electrochemical Society.**—The next general meeting will be held in Spring in Philadelphia, the exact date to be determined later.

**New York Section American Electrochemical Society.**—The first meeting of the section this season was held at the Chemists' Club on Dec. 11. There were some eighty members in attendance. Dr. Charles A. Doremus, as the retiring chairman of the Section, gave a review of the work of the Society. Mr. van Rensselaer Lansing then read a paper on the problems of electric lighting from the standpoint of the illuminating engineer. Prof. Samuel A. Tucker, of Columbia, was elected chairman of the Section for the coming year, Mr. Alois von Isakovics was re-elected as secretary and treasurer.

**American Association Advancement of Science.**—The New York meeting, held from Dec. 27 to 31 at Columbia University, was a full success. Jointly with Section C of the association, the American Chemical Society held its thirty-fifth general meeting. A long list of papers were presented in the sections for inorganic chemistry (H. L. Wells, chairman), organic chemistry (A. S. Wheeler), industrial chemistry (A. D. Little), agricultural and sanitary chemistry (L. L. van Slyke), physical chemistry (Alexander Smith), and biological chemistry (Wm. J. Gies). A special feature were two lectures on Niagara Falls, delivered at the new College of the City of New York. Dr. J. M. Clark gave a review of the legislation tending towards restriction of power development at Niagara, while Prof. C. F. Chandler spoke in a very interesting manner on the electrochemical industries of Niagara Falls, but especially on the aluminium industry. On the evening of Dec. 29 the Chemists' Club tendered a reception and smoker to the visiting chemists. On the same evening Dr. W. F. Hildebrand presented his address as president of the American Chemical Society, the subject being the present and future of the Society.

**Niagara Falls as an Industrial Center.**—We have received from Mr. Edward H. Taylor, of Niagara Falls, N. Y., a neatly illustrated pamphlet on the natural advantages of Niagara Falls as an industrial center. The booklet contains a great amount of interesting information and statistical data, especially on factory sites, power available, shipping facilities, selling advantages and Canadian trade. The pamphlet should be carefully read by all who are looking for new factory sites with great natural advantages.

**The Piqua Blower Co.**, of Piqua, Ohio, is being incorporated under the laws of Ohio with a capital of \$50,000. This corporation will take over the interests of the Piqua Foundry & Machine Co., Piqua, Ohio, and will make a specialty in the manufacture of positive blowers and gas exhausters as developed by the latter company in the past two years.

**Calendars.**—In the Roessler & Hasslacher Co's calendar for 1907 a well-known sentimental picture by Woennenberg has received quite an artistic reproduction. The calendar of the New York Brick Paving Co. of Syracuse, N. Y., will be refreshing to those who are tired of pretty girl pictures of the Howard Chandler Christy type; the picture on this calendar shows two boys, evidently full of mischief, but in the present case being contented with torturing a little fox terrier dog by means of a musical performance. A small calendar of the Crocker-Wheeler Co. gives a view of the works of this company.

**Advancing Market in Electrical Apparatus and Supplies.**—Prices in the electrical trade continue to show a distinct upward tendency in sympathy with the well maintained advance which has taken place in the prices of all raw materials. Orders for future delivery can only be placed in many instances at a considerable advance over present market quotations. The General Electric Company, in common with many other large manufacturing concerns, is announcing a general advance in prices of electrical apparatus and supplies. This will not unlikely be followed by further advances if present market conditions continue.

**The Goldschmidt Chemical Co.** has been formed for the purpose of taking over certain chemical processes and patents, other than their well-known thermit process, of Messrs. Th. Goldschmidt, Essen-Ruhr, Germany, and of manufacturing under them. Dr. Karl Goldschmidt is the president, Dr. Franz H. Hirschland the treasurer and manager of the company. Mr. E. Stütz and Mr. Hubert E. Rogers are vice-president and secretary respectively.

**Electric Steel in Germany.**—The Oberschlesische Eisenindustrie A. G., and the Poldihütte have decided to introduce the manufacture of tool steel in the electric furnace. The Kjellin induction furnace will be used and the installation will be made by the Siemens & Halske A. G.

### Personal.

Prof. J. J. THOMSON, of Cambridge, and Prof. HENRI MOISSAN, of Paris, received this year the Nobel prizes for physics and chemistry respectively. Prof. Thomson has specially distinguished himself through his renowned work in the evolution of the electronic theory, Prof. Moissan through his well-known high temperature research work with the electric furnace.

Mr. FRANCIS F. COLEMAN, formerly with the Westinghouse and Allis-Chalmers Companies' publicity departments and recently with the Traylor Engineering Co., has joined the Lidgerwood Manufacturing Co. as Publicity Manager. Mr. Coleman has also a host of friends from his former connections in an editorial capacity with the New York *Sun* and with *Electrical Age*.

### Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

#### CARBORUNDUM.

No. 492,767, Feb. 28, 1893, E. G. Acheson, of Monongahela City, Pa.

Produces a compound of the formula SiC, chemically silicide of carbon carbide of silicon, designated as "Carborundum," a crystalline substance, very brilliant, usually of a dark color, depending upon the materials used, the crystals being in general octohedral, although other well defined forms may be observed. Many of the crystals are opaque, some transparent and colorless, others of various colors. The crystals are extremely hard and highly refractory, withstanding, for a long time at least, an oxyhydrogen blast. The material, in its crystalline or powdered form, is a substitute for diamond powder and bort, for the cutting of diamonds and other gems, and is a suitable abrasive for optical purposes, dental work and grinding valves. It may be formed into wheels or discs, applied to cloth or paper or otherwise put into convenient form for use. An important use is for the filaments of incandescent lamps.

The product may be made by electrically heating a mixture of carbonaceous material, preferably carbon of high purity, with silica or its equivalent, aluminium or calcium silicate, in proper proportion, a flux, such as common salt, being preferably added. Coked bituminous coal or gas coke carbon, preferably of high purity, is properly sub-divided, mixed with silica or a silicate and a flux, such as common salt, the whole mass properly sub-divided and mixed, and the mixture subjected to a high degree of heat in an electric furnace, being placed in the furnace in such manner as to allow the electric current to pass through the mass. Owing to the relatively high initial resistance of the mass, it is sometimes desirable to provide a conducting core, conveniently of graphite, in a loose or solid condition. The furnace illustrated is a rectangular structure, having a flat bottom and vertical slide and end walls of fire-brick, electrodes extending through the end walls, a conducting core being shown between their ends, around which is the charge mixture, filling the furnace.

The specific charge consists by weight of pure carbon, 50 per cent; silica or aluminium silicate, 25 per cent; common salt, 25 per cent. An electric current is passed through the charge, or through the core, if used, the current being preferably increased as the operation proceeds. The charge is thereby highly heated, the flux is melted and the materials react to give the crystalline product. The product is generally found in a zone surrounding the core. The extent of this zone and the character of the crystalline product depend on the character of the charge, the strength of the current, time of operation, etc. During the process chemical combinations and decompositions take place, resulting in the production of gases, vapors and volatile salts, which escape, leaving in the furnace the crystalline product, by-products of graphite and amorphous carbon and a residue only partially converted, representing the various stages of the conversion, and which may form a portion of another charge. The products when removed from the furnace are separated, the carborundum is freed from the graphite and other impurities, broken into small pieces, which may be boiled, washed and dried, and finally be graded into different sized crystals by sifting, or powdered by stamp machines and separated into different grades of powder by floating in water. When the charge consists of carbon, silica and sodium chloride, the average composition of the carborundum is: Silicon, 69.19; carbon, 29.71; oxides of aluminium and iron, 0.38; calcium oxide, 0.19; magnesium oxide, 0.06; oxygen, 0.47.

When the charge consists of carbon, clay (aluminium silicate) and chloride of sodium, the product consists of silicon, 60.51; carbon, 30.09; oxides of aluminium and iron, 4.78; calcium oxide, 0.17; magnesium oxide, 9.18; oxygen, 0.27.

Reissue—No. 11,473, Feb. 26, 1895, E. G. Acheson, of Monongahela City, Pa.

This reissue is in general identical with the preceding patent, but states that it is preferable to work with a core and, specifically claims passing the electric current through a conducting core embedded in the charge.

